GOVERNMENT OF CANADA

DEPARTMENT OF ENERGY, MINES AND RESOURCES

CANADA CENTRE FOR MINERAL & ENERGY TECHNOLOGY

CANMET PROJECT 43102

IDENTIFICATION OF REQUIREMENTS FOR DEVELOPMENT (PLANNING & DESIGN) THAT ARE INFLUENCED BY THE NATURE & SIZE OF EQUIPMENT DEVELOPMENT (EXISTING AND PLANNED)

FINAL REPORT

(PHASE I)

MARCH 1982

MONTREAL ENGINEERING COMPANY, LIMITED

A MONENCO COMPANY

REQ. NO. 0-9085

CONTRACT REPORT NO. OSQ80-00185

IDENTIFICATION OF THE REQUIREMENTS FOR MINE DEVELOPMENT (PLANNING & DESIGN) THAT ARE INFLUENCED BY THE NATURE AND SIZE OF EQUIPMENT DEVELOPMENT (EXISTING & PLANNED)

PRELIMINARY REPORT

BY

MONTREAL ENGINEERING COMPANY, LIMITED

Work on this project was conducted under the auspices of the Canada Centre for Mineral and Energy Technology, Energy, Mines and Resources, Canada.

DSS FILE NUMBER : 23SQ.23440-0-9085
DSS CONTRACT SERIAL NUMBER : 0SQ80-00185
CANMET PROJECT NUMBER : 431002
CANMET SCIENTIFIC AUTHORITY : Dr. J. Pathak

217pp

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17 May 1982

Dr. J. Pathak
Department of Energy, Mines & Resources
CANMET Mining Research Laboratories
Bells Corner Complex
555 Booth Street
Ottawa Ontario K1A 0G1

Dear Sir:

RE: Identification of the Requirements for Mine Development (Planning and Design) that are Influenced by the Nature and Size of Equipment Development (Existing and Planned)
Canmet Project 43102 - Ref. No. 23SQ 23440-0-9085

We have pleasure in submitting our final report for Phase I of the above project, in fifteen bound copies and ten microfiche copies.

The report, which includes the comments made during the review of the draft report, contains seven parts.

Part 1 provides the background to the study; Part 2 deals with the study methodology; Part 3 gives the results of the mine equipment survey and analyses the results; Part 4 reports on the trends in the development of hard rock mining equipment and its adoption in Canadian Mines; Part 5 gives case histories of these developments; Part 6 records the conclusions and Part 7 presents the recommendations for Pase II of the study. Two Appendices complete the report.

We have found this first phase of the project extremely interesting and trust that this Phase I report will be to your satisfaction. We shall be pleased upon receipt of your approval, to prepare proposals for undertaking Phase II of the study in accordance with the recommendations made in this Phase I report.

We shall be pleased to answer any questions arising from the report, and look forward to our continuing association with the next phase.

Yours very truly,

Montreal Engineering Company, Limited

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MONTREAL ENGINEERING
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INTRODUCTION

1.0 INTRODUCTION

This study was authorized on February 13, 1981 by the Canada Department of Supply and Services on behalf of the Department of Energy, Mines and Resources, CANMET Mining Research Laboratories.

The authorizaion to proceed was based on the terms defined in a proposal from Montreal Engineering Company, Limited, dated 05 December 1980, in which the scope of the work was described as follows:

1. To identify the innovative developments which have recently taken place and are currently taking place in mining equipment for use, worldwide, in underground hard rock mining.
2. To produce case histories for each development.
3. To assess the potential application of each development to underground hard rock mining in steep and slightly inclined ore bodies in Canada. This assessment would include the likelihood, timing and extent of widespread adoption in Canada.
4. To produce briefs for each system assessing the critical engineering and economic factors, their effect on mine layouts, and in particular, their effect on safety, environment and ground control.

The study was separated into two parts. Part I comprising items 1, 2 and 3, and Part II item 4.

This project was assigned to the Mining Division of Montreal Engineering Company, Limited under the direction of Mr. I.R. Muirhead, Project Manager - Underground Mining.

We wish to acknowledge the help and encouragement which we have received from:

Dr. J. Pathak
Scientific Authority for the Project
Department of Energy, Mines and Resources
CANMET Mining Research Laboratories
Bells Corner Complex
555 Booth St.
Ottawa

and the many mining companies and equipment suppliers in Canada, U.S.A. and other countries, from whom much of the basic data has been received. A list of the contributing companies is given in Appendix B.

We also wish to acknowledge the information which has been obtained from the multitude of papers published in the technical press.

PART 2

STUDY METHODOLOGY

PART 2 - STUDY METHODOLOGY

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2.0 STUDY METHODOLOGY

Operations involving mining equipment can be broadly classed into six major areas - Drilling, Loading, Transportation, Support, Mechanized Excavation and Safety and Control.

Most of the recent developments in underground hard rock mining equipment have taken place within these fields.

It was therefore decided to seek information from mining companies, shaft sinking and tunnelling contractors, and equipment suppliers relating to recent developments, equipment currently in use, and the most likely developments in the future.

A questionnaire was devised (Appendix A) and issued as described in Section 3.2. In addition, literature searches were carried put using the following indices -

EMR CANMET - MINTECH
Engineering Index
N.T.I.S.

to obtain published data relating to each of the above fields.

The assembled data was then used to:

(i) assess the 'state of the art' in mining equipment usage in underground hard rock mining

(ii) assess the probable developments in the future

(iii) produce, where possible, brief case histories, and;

(iv) assess the likely potential for the new developments in Canadian hard rock mining.

The results from the questionnaire and searches are discussed in detail in the parts that follow.

PART 3

MINE EQUIPMENT DEVELOPMENT SURVEY ANALYSIS

PART 3 - MINE EQUIPMENT DEVELOPMENT - SURVEY ANALYSIS

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3.1 METAL MINING IN CANADA

Canada is one of the leading producers of metals in the world, ranking with the USSR, USA and Australia, a position which it has maintained for many years. Table 3-1 illustrates the changes that have taken place between 1970 and 1979 (latest available statistics). It will be noted that Canada has maintained its position in most metals, and appears to be gaining ground in the production of magnesium and tungsten, but is losing ground in silver and nickel production.

The tonnage of ore mined has been increasing steadily year by year but the mineral produced has not risen accordingly (Table 3-2). In fact, if one excludes the production of iron ore (because it constitutes approximately 95% of the annual metal output and its output is subject to the wide fluctuations in demand for steel), it can be seen from Table 3-3 that the production of other metals is in fact falling.

A comparison of tonnages for mineral production and ore mined illustrates very clearly (Table 3-2) that ore grades and recovery efficiencies are dropping quite drastically.

Table 3-4 gives the value of the mineral production (metals), together with the capital expenditure undertaken in metal mines for the same period. This latter figure includes exploration, construction, machinery and equipment. When this is taken as a percentage of the production value, further disturbing figures arise. They show a general tendency for the percentage of the value of mineral production spent as capital for new and expanding mines to have dropped over the past 10 years or so. Since the expenditure of capital now will generally not have an effect on production until several years hence, the picture is not a happy one. Taken with the fall in grades, etc., it looks even less promising.

As can be seen from Table 3-5, the number of producing metal mines has remained virtually constant over the last 10 years, although there has been a slight trend towards an increase in the percentage of surface mines (24%-31%) and a reduction in the percentage of underground mines (76%-69%). It will also be noted from Table 3-5 that the capacities of surface mines are on average five times those of the underground mines.

Finally in Table 3-6, details are given of the working time lost through strikes and lock-outs. As might be expected, this varies considerably, being largely dependent on when contracts are due for re-negotiation. 1969, 1975, 1978 and 1979 were particularly bad years. This obviously affects production and in 1975 labour disputes seemed to contribute to the closure of a number of underground mines.

3.2 TRENDS IN MINING, ECONOMICS AND EQUIPMENT

The trends in the hard rock mining industry are affected by many factors - political, economic, social and environmental, of which perhaps the economic aspect is the most important. Equally, the effects of federal and provincial policies and differences, and the general state of world trade can also have a significant effect.

As standards of living rise throughout the world, so does the demand for products, creating a corresponding increase in the demand for the basic mining commodities.

The majority of mining engineers will agree that most of the easily accessible and readily workable ore bodies have been found and that the ore bodies of tomorrow will be more difficult to find and tougher to work.

Similarly the higher grade ore bodies are becoming a thing of the past.

We are therefore faced with the need to produce more from leaner deposits situated in more inaccessible terrain and in more difficult mining conditions, and probably because of environmental and other pressures, only permissible as underground mines.

It is perhaps just as well that over the last 10-15 years, underground mining has changed from an art to more of a highly efficient science and technology.

Let us briefly look at some of the critical areas.

321 COST OF CAPITAL

With the current high cost of financing capital and the additional delays being created by governmental and environmental restrictions which tend to lengthen the time taken to bring a mine into production, it is becoming increasingly difficult to finance the development of new mines.

Aside from any steps that can be taken to reduce the 'regulatory and approval' time - a matter outside the scope of this report - one of the major means to reduce the cost of the mine development is to reduce the time to reach the production stage.

Possibilities exist in such fields as shaft boring and continuous or higher speed access and development drivage whereby the pre-production time can be reduced.

322 OPERATING COSTS

Operating costs have risen astronomically over the past 10-15 years. Labour costs have risen over 300% in the past 15 years, fuel costs by 280% in the past 6 years, and although the capital costs of mining equipment are high, their operating costs (excluding labour and fuel) have not risen at the same rate.

Taken with the dwindling skilled labour force and the need to increase productivity to mitigate these rising costs, the need to rethink present conventional mining methods is evident.

The rising costs and reducing labour force are dictating a trend to larger and more mechanical and energy efficient operations. This trend includes the more efficient use of manpower particularly in support services, the reduction of travelling time to the working face, electrification and the improvement of equipment and technology aided by the increasing use of computers and micro-processors.

323 ENVIRONMENT

One of the major problems facing the mining industry today is the mushrooming demand for environmental protection, particularly at potential surface mining operations.

Regulations for the operation of surface mines are becoming more stringent. These will of necessity lead to large increases in the tonnages of ore that will have to be produced from underground mines.

Underground mining is less environmentally disruptive, which should help to reduce the approval period, compared to surface mining.

324 MINING METHODS

The trend in underground mining is to have to mine deeper and lower grade ore bodies. At present, the major output from underground mines is from ore bodies less than 1200 m deep, but this will have to change in the future.

The pressure on costs is tending to push mining methods towards larger operations including the introduction of 'quarrying' methods underground (e.g. Vertical Crater Retreating).

Mining operations will increase in size until the optimum economic performance level is reached. This will involve the use of innovative techniques and methods, more efficient mine planning, extraction and recovery, enabling the recovery, for example, of ore formerly left as pillars or because of contamination. Improved roof control will be an essential part of this change.

A corollary to this trend to mine lower grade ore bodies is, of course, the development of the necessary techniques for extracting the metal from the ore at economic yields or recoveries, especially in secondary recovery techniques.

Since underground mining is less flexible than surface mining especially with the tendency to concentrate the production from fewer working places, there is a greater need to mechanize and automate the systems to maintain more efficient and effective control of the operations. Additionally, benefits can be increased by reducing the cyclical nature of the hard rock mining methods by applying continuously operating systems, for example, continuous loading and conveying.

New mining methods are being developed to take advantage of new techniques and innovations, aiming at reducing waste rock quantities and reducing the ratio of development and production (e.g.,large diameter blasthole stoping and vertical crater retreat mining).

325 EQUIPMENT

Mining methods are often limited by the machinery available yet they also often evolve as a direct result of newly introduced equipment. One has only to look at the great effect the introduction of the trackless vehicles had on mining methods in the '60's and '70's.

The process of developing new equipment (and technology) is slow and seems to be getting slower. Dramatic and sudden changes are rare. It often takes between 5 and 10 years for a new or unique technique to proceed from the initial idea to preliminary acceptance of the product by industry.

The normal process is one of re-design and upgrading to produce better, often larger and more productive machines.

New technology is also regrettably often dependent on the willingness of an operating company to 'stick its neck out' and try the ideas out underground. Most often it is the larger mine that provides the opportunity, yet the cost impact is frequently greater for the small mine where the ratio of labour cost to total cost is greater.

Mechanization of mining operations often alters the labour emphasis from skilled miners to skilled mechanics and electricians.

Recent trends are towards the use of hydraulic drilling, down the hole drills, continuous loading, electric LHD, conveyors, remote control, new support systems, electronics and the monitoring of operations, automated haulage systems, and shaft drilling techniques.

These trends will be examined in more detail in the sections following.

3.3 SURVEY OF MINES, CONTRACTORS AND SUPPLIERS

A questionnaire was issued to operating underground metal mines in an attempt to establish the type of mine, the mining methods being used, the mining equipment in use, and the potential for the development of mining equipment.

The questionnaire - a sample is given in Appendix A - was sent out to a total of 323 mines comprising:

(i) 96 mines in Canada (no production limit)
(ii) 62 mines in U.S.A. (production over 500 TPD)
(iii) 165 mines elsewhere in the world (production in excess of 1000 TPD).

The response was exceptionally good from the Canadian mines, average internationally and very poor from the U.S.A. (A normal response is considered to be between 20 and 25%.)

The results were:

	Issued	Replies	Percentage
Canada	96	55	57.3
U.S.A.	62	9	14.5
International	165	36	21.8
TOTAL	323	100	31.0%

In addition to the questionnaire, letters were sent out to (a) mining equipment suppliers (325) seeking details of the equipment available for use in hard rock mining, and of future development in this field, and (b) tunnelling and shaft sinking contractors in Canada and the U.S.A. (73) seeking information on equipment and techniques from those fields applicable to hard rock mining development and access needs.

The response was good (as might be expected) from the suppliers and average from the contractors.

	Issued	Replies	Percentage
Suppliers	325	156	48.0%
Contractors	73	16	21.9%

Overall, the total response was a satisfactory 37.7%.

3.4 ANALYSIS OF REPLIES

341 MINING COMPANIES

The information contained in the returned questionnaires has been summarized into several categories:

(i) For the 20 main equipment categories, the percentage usage in all mines, Canadian mines and International mines including the U.S.A. (EXHIBIT 3-1).

(ii) A breakdown of the equipment usage (by number of mine replies) against the type and size of mine, ore body characteristics and mining methods for all mines (EXHIBIT 3-2); International mines (EXHIBIT 3-3); and Canadian mines (EXHIBIT 3-4).

(iii) For the 20 main equipment categories, the percentage of replies indicating future potential, split into all mines, International and Canadian mines (EXHIBIT 3-5).

(iv) A breakdown of future potential for each equipment type against the type and size of mine, ore body characteristics and mining methods for all mines (EXHIBIT 3-6).

(v) Tabulation of the types of equipment which the mines considered had a considerable impact on mine design, layout, working methods, safety, ground control and economics. Results for all mines, International mines and Canadian mines are given in EXHIBIT 3-7.

A number of points had to be borne in mind when the results were analysed:

(i) Many questionnaires had not been fully completed, particularly on future potentials, thus the number of replies used for each category varied.

(ii) Many mines use more than one method of mining. Equipment usage was not identified to a particular method.

(iii) The number of replies received in each category was approximately the same from International mines and Canadian mines. Comparisons between the two could therefore be reasonably made.

(iv) Taking into account the degree of completion of questionnaires, it should be borne in mind that the equipment usage data comes from 25% of the world's major mines, and the potential usage data for only 14% of the major mines.

.1 Equipment Usage (EXHIBIT 3-1)

As might be expected, the most used categories overall were mechanized drilling equipment, belt conveyors and mechanized transport systems, DTH drills, low profile loading equipment and raise borers.

However, quite a different picture emerges when International usage and Canadian usages are compared. The most obvious differences are:

- Canadian mines prefer low profile equipment, mobile crushers (mainly potash) and to a lesser extent DTH and mechanized drilling equipment, belt conveyors and the new support systems.
- International mines use more mechanized transport systems, raise borers, Haggloaders and the pipelining of ore and fill. Although a difference is shown in the use of 'tunnelling' machines, this actually reflects attempts to use roadheader type machines (rocks too hard) plus the use of continuous miners in potash.

.2 Mining Methods and Equipment (EXHIBIT 3-2; 3-3; 3-4)

The trends indicated in Exhibit 3-2 for all mines are similar to those developed from Exhibit 3-1, namely drilling, transportation and loading. It will be seen that the bulk of mines have outputs in excess of 1000 TPD and have steep massive or veined ore bodies. The most common mining methods are open stoping, cut and fill, sublevel blasthole and room and pillar. Equipment usage (as a percentage of the number of mines using each mining method) is relatively consistent for mechanized drilling for all methods. If only the methods applicable to steeper ore bodies are considered, then equipment such DTH drills, boring, mechanized transport, conveyors and the usage of low profile equipment appears to have reasonably common application.

It is, however, of more interest to compare the separate figures for the Canadian and International mines (EXHIBITS 3-3 and 3-4). A number of differences are readily evident:

(i) International mines have larger capacities though ore body configurations are similar. This is particularly true in Africa.

(ii) Canadian mines appear to be spearheading the use of 'mass' blast methods utilizing DTH drills - namely sublevel blasthole, vertical crater retreat and large diameter blasthole methods (32 mines to 12). Similarly, more use is made by Canadian mines of the shrinkage stoping, open stoping and cut and fill methods (38 to 25).

International mines still favour the block caving and sublevel caving methods (15 to 4)

(iii) Looking at equipment usages, Canadian mines make more use of DTH drills, mechanized drilling systems and conveyors, whilst the International mines make more use of raise borers and mechanized transport systems (excluding LHD).

.3 Equipment with Development Potential (EXHIBIT 3-5)

Only 44% of the mines who that returned the questionnaire completed the section relating to equipment potential. The results are expressed in Exhibit 3-5.

Viewed as a whole, the general concensus is that there is development potential in all drilling equipment, transport systems and conveyors, remote control, low profile equipment, raise boring and new support systems.

However, there is a distinct difference in emphasis between the International and Canadian viewpoints. The only areas of equal emphasis are for monitoring, raise boring and pipeline transport. The International mines exclusively favour blind boring, whereas the Canadians exclusively favour portable crushers and tunnelling equipment.

However, the Canadians place much more emphasis than the Internationals on drilling, loading, safety, remote control and support.

Exhibit 3-6 shows the trends in equipment potential related to the numbers of mines in each category (size, ore body, method). This generally bears out the comments above.

.4 Factors Affected by Equipment Type (EXHIBIT 3-7)

Attempts were made with the questionnaire to identify which types of equipment were considered to have an impact on certain factors affecting mining operations. Sixty-nine respondents completed this section - 33 Canadian, 36 International.

In Exhibit 3-7, the positions of the equipment types having the more important impact have been tabulated for all mines, International mines and Canadian mines.

Again, differences are clearly seen between Canadian and International interpretations in each of the six categories - mine design, layout, working method, safety, ground control and economics - and generally conform to the differences in mining methods and equipment usages noted earlier.

The changes in emphasis are perhaps better illustrated in Table 3-7, which shows the relative importance of each equipment type based on its standing in each category. The three clear distinctions shown are the importance given to raise borers and safety equipment Internationally and to remote control in Canada.

342 SHAFT AND TUNNELLING CONTRACTORS

A total of 16 replies were received in answer to the 73 letters sent to shaft sinking and tunnelling contractors in Canada and the U.S.A. Of these, only five were able to offer constructive comment.

These comments relate to the successful use of tunnelling machines, drilling equipment, crushers and conveyors, support systems, shaft drilling, and pipeline and rail transport of rock, in tunnel drivages and shaft sinking.

The comments supplied indicate that the mining industry could benefit considerably from the experiences of the contractors in the areas listed above. These comments have been incorporated in the descriptive Parts 4 and 5.

343 MANUFACTURERS AND SUPPLIERS

The replies received from manufacturers and suppliers have mainly taken the form of brochures and leaflets. A number did include data on the operation of their equipment in mines and this data has been incorporated into Parts 4 and 5 following.

TABLE 3-1

Canada's Position as a World Metal Producer

	YEAR									
	70	71	72	73	74	75	76	77	78	79
NICKEL	1	1	1	1	1	1	1	1	2	2
SILVER	2	1	2	1	1	2	3	3	3	5
GOLD	3	2	3	3	3	3	3	3	3	3
PLATINUM	3	3	3	3	3	3	3	3	3	3
ZINC	1	1	1	1	1	1	1	1	1	1
COPPER	4	4	3	3	4	4	4	4	4	4
LEAD	5	5	4	4	4	4	4	4	4	4
COBALT	3	3	5	6	7	9	7	8	6	10
TUNGSTEN	6	7	7	7	7	9	8	8	7	5
MAGNESIUM	5	8	8	8	8	8	8	7	7	6
IRON ORE	?	5	6	6	5	7	6	5	7	6

TABLE 3-2

Canada - Ore Mined and Mineral Produced (Metal Mines)

YEAR	METALLIC ORE MINED (MILLION TONNES)	MINERAL PRODUCTION (MILLION TONNES)	AVERAGE RECOVERY (%)	TONNES MINED PER MHR PAID
1980	N.A.	52.91	N.A.	N.A.
1979	274.8	62.46	22.47	3.23
1978	248.1	45.13	18.19	2.92
1977	299.5	54.14	18.07	2.97
1976	296.5	57.65	19.44	2.95
1975	264.2	51.8	19.61	2.58
1974	278.7	54.78	19.65	2.68
1973	274.7	57.80	21.04	2.79
1972	205.9	46.19	21.85	2.20
1971	211.4	49.76	23.36	2.07
1970	213.0	50.75	23.83	1.97
1969	172.0	38.47	22.37	1.80
1968	186.9	45.33	24.25	1.78

TABLE 3-3

Canadian Metal* Production (Million Tonnes)
(Excluding Iron Ore)

YEAR	PRODUCTION
1980	2.04
1979	2.28
1978	2.20
1977	2.39
1976	2.23
1975	2.42
1974	2.56
1973	2.69
1972	2.48
1971	2.41
1970	2.38
1969	2.11
1968	2.20

* Zn, Cu, Pb, Ni, Mo, Mg, Co, Cd, Ag, Au, Sb

TABLE 3-4

Value of Canadian Metal Production and Capital Expenditure (Metal Mines) (Million Canadian $)

YEAR	VALUE OF PRODUCTION	CAPITAL EXPENDITURE		TONNES MINED PER MANHOUR
		TOTAL*	% OF PRODUCTION	
1980	9557	1451.2	15.2	N.A.
1979	8000	884.3	11.0	3.23
1978	5698	567.2	10.0	2.92
1977	5988	967.2	16.2	2.96
1976	5073	893.3	17.6	2.95
1975	4794	710.3	14.8	2.58
1974	4853	562.3	11.6	2.68
1973	3793	594.1	15.7	2.79
1972	2946	655.1	22.2	2.20
1971	2940	826.9	28.0	2.07
1970	3115	481.0	15.4	1.97
1969	2717	380.0	14.0	1.80
1968	2981	443.0	14.9	1.78

* Exploration, Construction, Machinery & Equipment

TABLE 3-5

Number of Operating Metal Mines in Canada

YEAR	SURFACE	UNDERGROUND	TOTAL
1980	52	113	165
1979	52	112	167
1978	45	120	165
1977	47	115	162
1976	47	114	161
1975	49	114	163
1974	47	127	174
1973	47	122	169
1972	40	125	165
1971	40	125	165

NOTE: In 1980, the capacities of the various mines were:

	SURFACE	UNDERGROUND
TOTAL CAPACITY (TPD)	829850	369080
AVERAGE CAPACITY (TPD) - EACH MINE	15950	3295
AVERAGE CAPACITY (TPD) - RANGE	100 - 135000	100 - 35000

TABLE 3-6

Time Lost Through Strikes/Lockouts
Canadian Mines

YEAR	MANDAYS LOST ('000)	MANDAYS WORKED ('000)	%
1980	418	N.A.	N.A.
1979	1522	10640	14.30
1978	1699	10613	16.00
1977	91	12650	0.72
1976	579	12575	4.60
1975	1179	12800	9.21
1974	515	13000	3.96
1973	221	12300	1.80
1972	335	11725	2.86
1971	194	12763	1.52
1970	54	13525	0.40
1969	2088	11975	17.44
1968	101	13150	0.77

MONTREAL ENGINEERING COMPANY, LIMITED

EXHIBIT 3-1
SURVEY RESULTS
EQUIPMENT IN USE UNDERGROUND

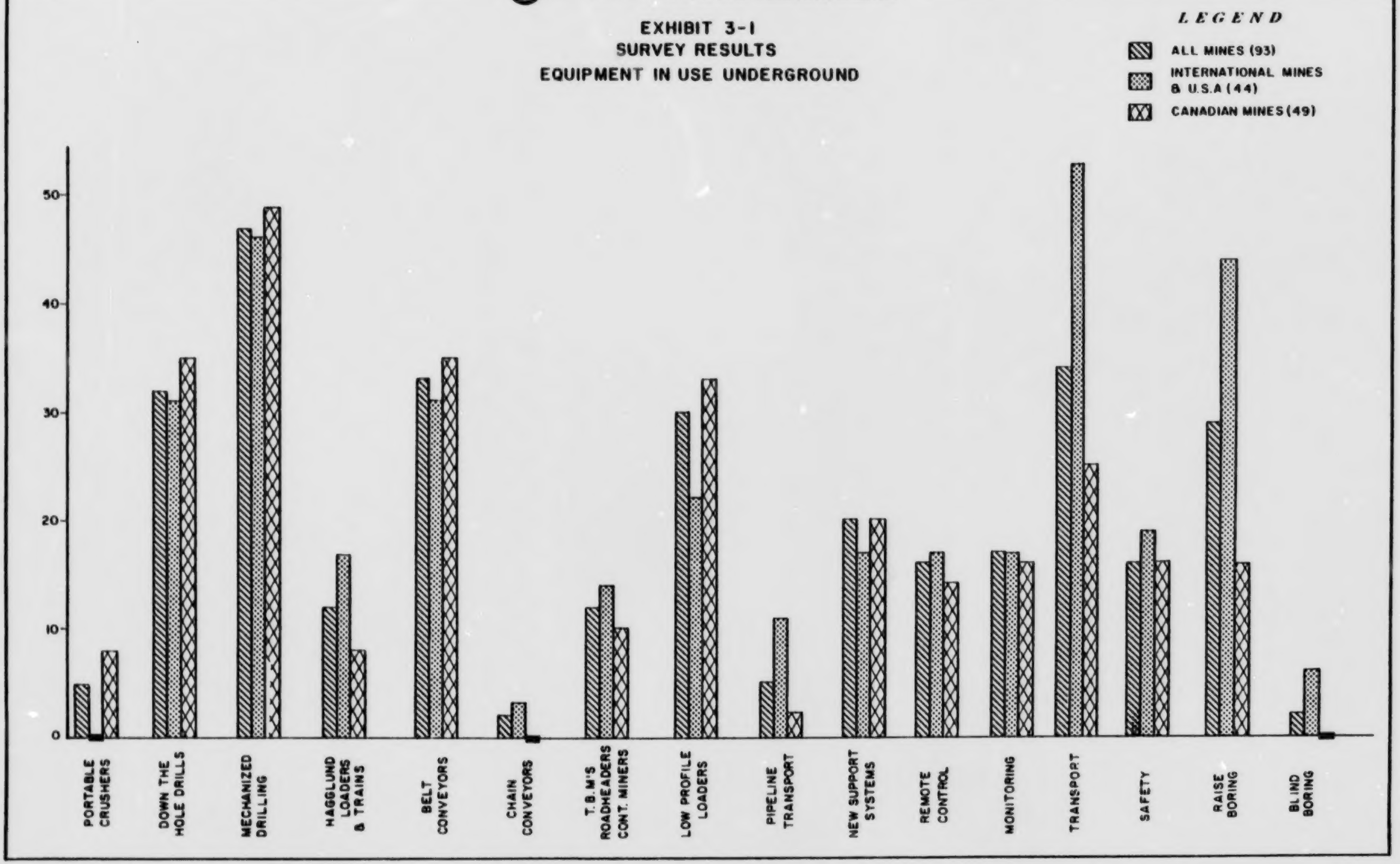

EXHIBIT 3-2

HARD ROCK MINING EQUIPMENT

IN USE - UNDERGROUND

ALL MINES (77)

No. of Mines
⬤ 16 - 20
● 11 - 15
• 6 - 10
· 1 - 5

MINING EQUIPMENT TYPES		Size of Mine: <500 TPD	500 - 1000 TPD	1000 - 5000 TPD	>5000 TPD	Type of Ore Body: Massive	Veined	Complex	>45°	45° - 10°	< 10°	Methods of Mining: Room & Pillar	Block Caving	Panel Caving	Sublevel Caving	Shrinkage Stoping	Open Stoping	Cut & Fill	Undercut & Fill	Sublevel Blasthole	Vertical Retreat Blasthole	Large dia. Blasthole	Benching	Top Slicing	Square Set	Other
# of Mines in Each Category		14	30	84	95																					
Data Given By		8	8	30	35	34	29	21	43	22	13	20	7	-	12	12	26	25	4	24	8	12	5	4	5	9
A. Portable Crushers	6%	-	-	1	4	2	3	1	-	1	3	·						·	·		·	·				·
B. In the Hole Drills	44%	3	2	13	16	17	10	10	23	8	3	·	·		·	•	●	●	·	●	•	●	·	·	·	•
C. Jumbo or Hydraulic	58%	3	6	20	16	24	14	9	26	10	4	●	·		•	•	●	⬤	·	●	•	•	·	·	·	·
D/E. Haggland loaders/trains	16%	1	1	2	8	9	1	4	7	2	1	·	·		·	·	·	·	·	·	·	·		·	·	·
F. Belt Conveyors	40%	1	1	9	20	12	10	9	16	7	9	●	·		•	·	•	•	·	•	·	·	·	·	·	·
I. Fullface tunnel boring machines	1%																·									
J. Roadheader type machines (* see note 1)	5%	-	-	-	11	4	6	-	1	-	8	·					·	·		·		·				·
K. Continuous Miners	5%											·					·									
L. Low profile equipment	31%	4	3	11	16	6	9	10	11	9	3	●	·		•	·	•	•	·	•	·	·	·	·	·	·
M. Pipeline transport of ore, etc.	6%	-	-	3	2	2	1	2	3	1	-	·			·		·	·	·	·	·	·				
N. New support systems	25%	1	1	5	12	8	6	6	12	9	4	•	·		·	·	•	•	·	·	·	·	·	·	·	·
P. Remote control of equipment	18%	-	1	5	8	6	4	7	9	3	-	·	·		·	·	·	•	·	•	·	·		·	·	·
Q. Monitoring equipment	21%	-	1	5	10	8	4	6	10	7	4	·	·		·	·	•	•	·	·	·	·	·	·		
R. Transport ore, men, materials	40%	2	3	7	19	16	9	6	14	10	5	•	·		•	·	•	•	·	•	·	·	·	·	·	·
S. Safety & Environment	21%	2	1	5	8	7	6	4	10	6	1	•	·		·	·	•	•	·	·	·	·	·	·	·	·
T. Raise Boring	38%	2	2	10	15	12	9	9	17	10	3	·	·		•	·	●	●	·	•	·	·	·	·	·	·
U. Blind Shaft boring	3%	-	-	1	1	-	1	-	1	1	-				·		·	·		·						

* Note 1 - 5 mines have tried these machines but were not successful. Rock too hard.

EXHIBIT 3-3

HARD ROCK MINING EQUIPMENT

IN USE - UNDERGROUND

INTERNATIONAL MINES (38)

No. of Mines
⬤ 16 - 20
● 11 - 15
• 6 - 10
· 1 - 5

		Size of Mine					Type of Ore Body						Methods of Mining														
		<500 TPD	500 - 1000 TPD	1000 - 5000 TPD	>5000 TPD		Massive	Veined	Complex	< 45°	45° - 10°	< 10°	Room & Pillar	Block Caving	Panel Caving	Sublevel Caving	Shrinkage Stoping	Open Stoping	Cut & Fill	Undercut & Fill	Sublevel Blasthole	Vertical Retreat Blasthole	Large dia. Blasthole	Benching	Top Slicing	Square Set	Other
# of mines in each Category		4	17	129	77																						
Data given by		4	--	14	22		18	13	9	17	12	6	8	7	--	8	2	14	9	1	9	1	2	2	2	2	6
MINING EQUIPMENT TYPES																											
A. Portable Crushers	3%	--	--	1	--	1	1	--	--	--	--	1	•														
B. In the Hole Drills	39%	1	--	4	10	15	6	6	4	9	4	2	•	•		•		●	●	•	•	•	•				•
C. Jumbo or Hydraulic Drilling	50%	1	--	10	8	19	12	5	2	10	4	1	•	•		•	•	•	●		●		•	•	•	•	•
D-E. Hagglund loaders/trains	21%	1	--	1	6	8	6	1	1	4	1	--		•		•	•	•	•		•					•	•
F. Belt Conveyors	32%	--	--	3	9	12	4	3	5	5	4	5	•	•		•		•	•		•			•	•		•
I. Full face tunnel boring machines	3%																	•									
J. Roadheader type machines (*see note 1)	5%	--	--	--	5	5	3	1	--	1	--	2	•					•	•		•		•				•
K. Continuous Miners	8%												•														
L. Low profile equipment	32%	3	--	6	3	12	3	4	5	5	3	1															
M. Pipeline transport of ore, etc.	11%	--	--	3	1	4	1	1	1	3	1	--	•			•		•	•	•	•						
N. New support systems	24%	1	--	2	6	9	3	3	3	5	5	2	•	•		•	•	•	•		•			•	•	•	•
P. Remote control of equipment	18%	--	--	3	4	7	3	2	3	3	2	--	•	•		•		•	•		•		•				•
Q. Monitoring equipment	21%	--	--	3	5	8	3	2	4	4	3	2	•	•		•		•	•		•		•	•	•		
R. Transport ore, men, materials	50%	1	--	5	13	19	10	6	4	8	8	2	•	•		•	•	●	•	•	•						•
S. Safety & Environment	18%	1	--	3	3	7	2	4	3	5	4	--	•	•		•	•	•	•		•					•	•
T. Raise Boring	55%	2	--	7	12	21	8	7	6	9	9	2	•	•		•	•	●	●		•	•		•	•	•	•
U. Blind Shaft boring	5%	--	--	1	1	2	--	1	--	1	1	--				•		•	•		•						

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*Note 1 - 3 mines have tried these machines
but were not successful. Rock too hard.

No. of Mines
⬤ 16 - 20
● 11 - 15
• 6 - 10
· 1 - 5

EXHIBIT 3-4

HARD ROCK MINING EQUIPMENT
IN USE - UNDERGROUND

CANADIAN MINES (39)

		Size of Mine				Type of Ore Body						Methods of Mining														
		<500 TPD	500 - 1000 TPD	1000 - 5000 TPD	>5000 TPD	Massive	Veined	Complex	>45°	45° - 10°	<10°	Room & Pillar	Block Caving	Panel Caving	Sublevel Caving	Shrinkage Stoping	Open Stoping	Cut & Fill	Undercut & Fill	Sublevel Blasthole	Vertical Retreat Blasthole	Large dia. Blasthole	Benching	Top Slicing	Square Set	Other
# of Mines in Each Category		14	13	51	18																					
Data given by		4	8	16	13	16	16	12	26	10	7	12	--	--	4	10	12	16	3	15	7	10	3	2	3	3
MINING EQUIPMENT TYPES																										
A. Portable Crushers	10%	--	--	--	4	1	3	1	--	1	2	·						·	·		·	·				
B. In the Hole Drills	49%	2	2	9	6	11	4	6	14	4	1	·			·	•	•	•	·		•	•	·	·	·	·
C. Jumbo or Hydraulic Drilling	67%	2	6	10	8	12	9	7	16	6	3	•			·	•	•		·	•	•	•	·	·	·	·
D-E. Hagglund loaders/trains	10%	--	1	1	2	3	--	3	3	1	1	·			·	·	·	·	·	·	·	·		·		·
F. Belt Conveyors	49%	1	1	6	11	8	7	4	11	3	4	•			·	·	·	·	·	·	·	·	·	·	·	·
I. Full face tunnel boring machines	---																									
J. Roadheader type machines (*see note 1)	5%	--	--	--	6	1	5	--	--	--	6	·														
K. Continuous Miners	10%											·														
L. Low profile equipment	31%	1	3	5	3	3	5	5	6	6	2	·			·	·	·	·	·	·	·	·	·	·		
M. Pipeline transport of ore, etc.	3%	--	--	--	1	1	--	1	--	--	--	·						·		·	·	·				
N. New support systems	26%	--	1	3	6	5	3	3	7	4	2	·			·	·	·	·	·	·	·	·	·	·		·
P. Remote control of equipment	18%	--	1	2	4	3	2	4	6	1	--	·			·	·	·	•	·	·	·	·		·	·	·
Q. Monitoring equipment	21%	--	1	2	5	5	2	2	6	4	2	·			·	·	·	·	·	·	·	·	·	·		
R. Transport ore, men, materials	31%	1	3	2	6	6	3	2	6	2	3	·			·	·	·	·	·	·	·	·	·	·		·
S. Safety & Environment	23%	1	1	2	5	5	2	1	5	2	1	·			·	·	·	·	·	·	·	·	·	·	·	·
T. Raise Boring	21%	--	2	3	3	4	2	3	8	1	1				·	·	·	·	·	·	·	·	·	·	·	·
U. Blind Shaft boring	--	--	--	--	--	--	--	--	--	--	--															

*Note 1. - 2 mines have tried these machines
but were not successful. Rock too hard.

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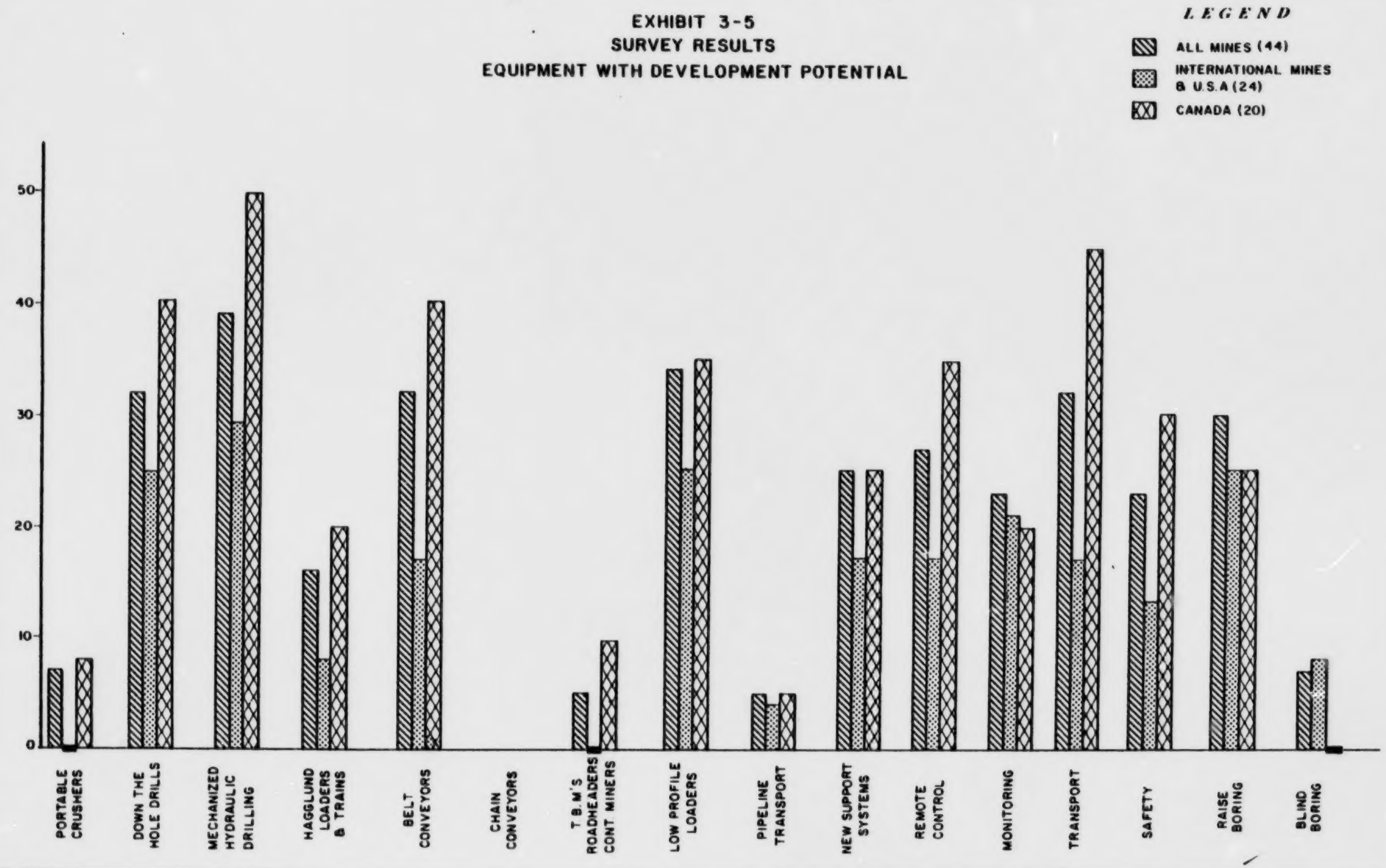
EXHIBIT 3-5
SURVEY RESULTS
EQUIPMENT WITH DEVELOPMENT POTENTIAL



TRANSLAR [illegible]

EXHIBIT 3-6

HARD ROCK MINING EQUIPMENT

POTENTIAL APPLICATION UNDERGROUND

ALL MINES (36)

No. of Mines
● 16 - 20
● 11 - 15
• 6 - 10
· 1 - 5

		Size of Mine				Type of Ore Body						Methods of Mining														
		< 500 TPD	500 - 1000 TPD	1000 - 5000 TPD	>5000 TPD	Massive	Veined	Complex	>45°	45°- 10°	< 10°	Room & Pillar	Block Caving	Panel Caving	Sublevel Caving	Shrinkage Stoping	Open Stoping	Cut & Fill	Undercut & Fill	Sublevel Blasthole	Vertical Retreat Blasthole	Large dia. Blasthole	Benching	Top Slicing	Square Set	Other
# of Mines In Each Category		14	30	184	95																					
Data Given By		3	4	10	19	16	9	11	20	9	7	11	4	-	8	6	9	13	4	9	6	6	2	3	1	2
MINING EQUIPMENT TYPES																										
A. Portable Crushers	3%	-	-	-	1	-	-	1	1	1	-						•	•			•					
B. In the Hole Drills	39%	2	-	4	8	5	3	6	9	4	1	•	•		•	•	●	●	•	•	•	•		•	•	•
C. Jumbo or Hydraulic Drilling	42%	1	3	5	6	8	3	4	11	4	-	•	•		•	•	•	●	•	•	•	•	•	•		•
D-E. Hagglund loaders/trains	17%	1	-	-	5	3	1	4	5	3	1	•			•	•	•	•	•	•	•	•		•		
F. Belt Conveyors	39%	1	-	1	12	8	1	4	5	3	6	●	•		•	•	•	•	•	•	•	•	•	•		•
I. Full face tunnel boring machines	6%	1	-	-	1	-	2	-	-	-	2	•				•	•			•		•				
J. Roadheader type machines	-	-	-	-	-	-	-	-	-	-	-															
K. Continuous Miners	-	-	-	-	-	-	-	-	-	-	-															
L. Low Profile equipment	31%	2	-	5	4	6	2	4	7	5	2	•	•		•	•	•	•	•	•	•	•	•	•		•
M. Pipeline transport of ore, etc.	3%	-	-	1	-	1	-	-	1	-	-								•							
N. New support systems	33%	1	1	3	7	4	1	5	6	5	1	•			•	•	●	•		•	•	•	•	•		•
P. Remote control of equipment	31%	1	1	4	5	5	1	5	9	2	1				•	•	•	●	•	•	•	•		•		
Q. Monitoring equipment	25%	1	-	2	6	5	1	4	7	2	1				•	•	•	•	•	•	•	•		•		
R. Transport ore, men, materials	42%	3	1	5	6	8	3	5	11	3	1	•	•		●	•	•	●	•	•	•	•		•		•
S. Safety & Environment	19%	-	-	1	6	5	-	4	6	2	-	•	•		•	•	•	•	•	•	•	•		•		•
T. Raise Boring	44%	3	1	4	8	7	4	6	10	4	1	•	•		●	•	•	●	•	•	•	•		•		•
U. Blind Shaft boring	11%	1	-	1	2	1	2	-	1	2	-	•			•	•	•	•								•

EXHIBIT 3.7

FACTORS CONSIDERABLY AFFECTED BY EQUIPMENT TYPE

Equipment considered to have an impact on: (in order of importance)

	MINE DESIGN	LAYOUT	WORKING METHODS
All Mines (69)	Down the Hole Drills = Raise Borers Electrohydraulic Drilling Low Profile Equipment = Belt Conveyors Transport Systems	Electrohydraulic Drilling D.T.H. Drills Transport Systems Low Profile Equipment Belt Conveyors = Raise Borers	Electrohydraulic Drilling D.T.H. Drills Transport Systems Low Profile Equipment Support Systems = Remote Control
International Mines (36)	Raise Borers D.T.H. Drills Electrohydraulic Drilling Low Profile Equipment = Transport Systems Belt Conveyors	Raise Borers = Transport Systems Electrohydraulic Drilling D.T.H. Drills = Low Profile Equipment Belt Conveyors	Electrohydraulic Drilling Transport Systems Haggloaders = Raise Borers D.T.H. Drills = Low Profile Equipment
Canadian Mines (33)	D.T.H. Drills = Electrohydraulic Drilling Conveyors Low Profile Equipment Haggloaders TBM's New Support Systems = Remote Control Transport Systems Raise Borers	D.T.H. Drills = Electrohydraulic Drilling Belt Conveyors Low Profile Equipment = Support Systems	Electrohydraulic Drilling D.T.H. Drills Low Profile Equipment New Support Systems = Remote Control Haggloaders

EXHIBIT 3.7 (Cont'd)

FACTORS CONSIDERABLY AFFECTED BY EQUIPMENT TYPE

Equipment considered to have an impact on: (in order of importance)

	SAFETY	GROUND CONTROL	ECONOMICS
All Mines (69)	Electrohydraulic Drilling	New Support Systems	Electrohydraulic Drilling
	Raise Borers	Monitoring	D.T.H. Drills =
	D.T.H. Drills	Electrohydraulic Drilling	Raise Borers
	Transport Systems	D.T.H. Drills	Transport Systems
	New Support Systems =	Support Systems	Support Systems
	Remote Control	Transport Systems	Low Profile Equipment
International Mines (36)	Raise Borers	New Support Systems	Raise Borers
	Electrohydraulic Drilling	Raise Borers	Electrohydraulic Drilling
	Transport Systems	D.T.H. Drills	Transport Systems
	D.T.H. Drills	Electrohydraulic Drilling	Low Profile Equipment =
	Remote Control =	Low Profile Equipment =	D.T.H. Drills
	Haggloaders	Monitoring	Conveyors
		Transport Systems	Support Systems =
			Monitoring
Canadian Mines (33)	Electrohydraulic Drilling	New Support Systems	Electrohydraulic Drilling
	D.T.H. Drills	Monitoring	D.T.H. Drills
	New Support Systems	Electrohydraulic Drilling	Support Systems
	Remote Control	D.T.H. Drills	Transport Systems
	Transport Systems =	Transport Systems	Conveyors
	Monitoring Systems	TBM's	Low Profile Equipment

PART 4

TRENDS IN THE DEVELOPMENT OF HARD ROCK MINING EQUIPMENT AND ITS ADOPTION IN CANADIAN MINES

PART 4 - TRENDS IN THE DEVELOPMENT OF HARD ROCK MINING EQUIPMENT AND ITS ADOPTION IN CANADIAN MINES

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4.1 GENERAL

The discussions that follow in this Section will be dealt with under six main headings, each subdivided as required.

An attempt is made to look briefy at the historical or general background to each major type leading to the current development trend. Where possible, assessments are made of the likely application of these developments to Canadian metal mining. Technical papers describing a typical example of the use of each type of equipment are given in Part 5.

The major and sub-division headings that will be considered are:

Drilling

- Development Drilling
- Stope Drilling

Loading

- Development
- Production

Transportation

- Ore
    - (a) Trackless loaders
    - (b) Truck
    - (c) Rail
    - (d) Conveyors and Mobile Breaker/feeders
    - (e) Pipeline
- Materials
    - (a) Trackless
    - (b) Rail
    - (c) Monorail
- Men
    - (a) Trackless
    - (b) Rail
    - (c) Monorail
    - (d) Rope
    - (e) Conveyor

Support

- Rock Bolts
- Cable Bolts
- Shotcrete
- Rock Bolting Mechanization
- Mechanization of Shotcrete applications.

Mechanized Drivage, Shaft Sinking and Raising

- Drivage

  (a) Continuous miners
  (b) Roadheaders
  (c) Tunnel boring machines (T.B.M's)

- Raising

  (a) Raise boring
  (b) Blind boring

- Shaft Sinking

  (a) Raise Borers
  (b) Shaft Borers using Pilot Holes
  (c) Blind Shaft Drilling Units
  (d) Blind Shaft Borers

Safety and Control

- Safety

  (a) Mechanized roof scaling
  (b) Impact breakers
  (c) Autoloading of explosives
  (d) Electric Detonators
  (e) Shaft Signalling Systems

- Control

  (a) Remote control of loaders
  (b) Automatic control of Drill Jumbos
  (c) Monitoring of Ventilation, Conveyors, Strata, etc.

4.2 DRILLING

421 HISTORICAL DEVELOPMENT

It is interesting to take a brief look at the development of underground drilling equipment for use in rock.

The earliest holes were formed by the rather brutal method of the hand held chisel and hammer. The first pneumatic drill is believed to have been developed by Sommellier in 1861. By 1910, the compressed air percussive rock drill (basically similar in principle to present day machines) was beginning to come into general use.

The 1930's saw general improvements in the percussive drill, but it was still limited to relatively short holes. However, diamond drill costs had become low enough for the equipment to be used for longhole drilling in stopes, and with adaptions, to radial and ring drilling, and the cutting of slots.

Around 1940, detachable steel bits and power feed mountings further increased drilling productivity and flexibility.

The introduction of tungsten carbide drilling bits after World War II finally made it economically and practically possible for longhole percussive drilling to take place in underground conditions. Improvements in alloy steels enabled greater striking power, drill rotation, and improved rod lives to be obtained. The availability of tungsten carbide also led to the introduction and extended use of airleg mountings and a move away from barmounted cradles to the simple forms of jumbo with manual adjustments and/or hand operated hydraulics.

The 50's and 60's saw the development of the fully mechanized compressed air percussive and rotary percussive drill rigs with hydraulically positioned booms.

In the late 60's and early 70's, the hydraulic drill started to make its appearance on the tunnelling scene in Europe, reaching North America in 1973, about the same time that down-the-hole drills were first being tried underground. It is, however, only in the past three or four years that hydraulic drilling and in-the-hole drilling has started to become more popular.

422 USES OF DRILLING EQUIPMENT UNDERGROUND

The applications for drilling equipment (excluding raise drilling which is the subject of a separate section) can be generally split into two categories.

- Development and Access drilling
- Stope drilling

.1 Development and Access Drilling

The equipment used is dependent on the heading size which in turn can depend on whether the heading is in ore or waste rock, the strength of the ore and rock, the size of the loading equipment, and the demands of services such as ventilation.

In large headings, multiboom rubber tire or crawler mounted jumbos are used. The number of booms range from two to five with the possible addition of working/charging platforms. Vehicle power units can be diesel or electric, with drills powered by compressed air or hydraulics. (Figs. 4-1 & 4-2)

.2 Stope Drilling

The type of drilling equipment used is largely governed by the mining method, although quite often the converse is true (e.g. VCR stoping evolved from the availability of DTH drills).

However, broadly speaking, the trends are as follows:

- In shrinkage stoping, square set and overhand stoping, airleg mounted jack hammers and stopers are used.

- In vertical crater retreat, sublevel retreat, and large diameter blasthole stoping systems, Down-the-Hole (DTH) or as they are sometimes call In-the-Hole (ITH) drill rigs are used.

- In sublevel stoping, sublevel caving and to a lesser extent block caving, longhole drilling rigs are used including fan and ring drilling versions.

- In room and pillar and cut and fill stoping, drill jumbos and light mechanized wagon drills are usually used.

Drilling jumbos, specially adapted, are also used for rock bolting. These are dealt with in the section on support.

423 RECENT DEVELOPMENT AND TRENDS

It is impossible to detail all the recent developments and trends in drilling equipment that have occurred in the past 10 years, since they would take up a volume on their own. However, based on the comments made in the questionnaires, and from recent articles in the technical press, the more important features can be listed as:

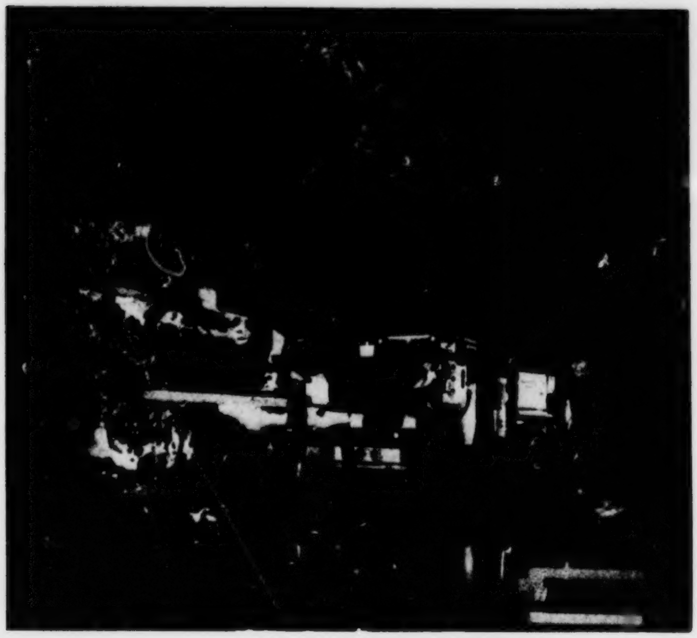


Fig. 4-1
Multiboom electrohydraulic drilling rig
in a development heading
(Courtesy Atlas Copco)

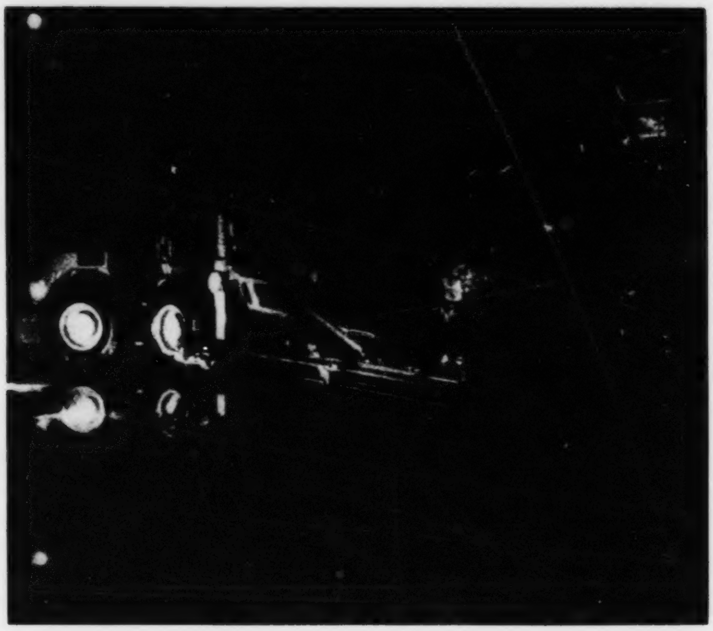


Fig. 4-2
Hydraulic drill jumbo
for small headings
(Courtesy Tamrock)

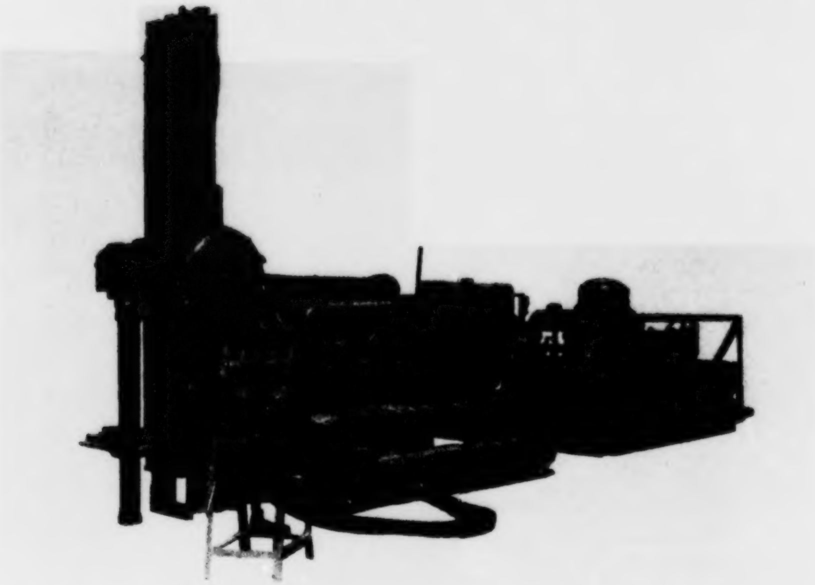


Fig. 4-3
In the hole (ITH) drill with
skid-mounted booster compressor
(Courtesy TRW Mission)

(i) (a) Electric hydraulic drilling with Jumbo, Fan, Ring, Longhole, and Hand methods;
    (b) Automatic collaring, retraction, anti-jamming;

(ii) In-the-hole (ITH) drilling for down, up, inclined operations; and

(iii) Fully automatic computer controlled drilling.

.1 Electro-Hydraulic Drilling

An electro-hydraulic drilling is defined as a hydraulic drill with drilling power supplied by electrically driven pumps, although diesel power is often retained to move the drill vehicle between working faces. Hydraulic drills have been described in detail in many excellent papers. It is sufficient to state that hydraulic fluid instead of compressed air provides the percussive energy to the drill and drill rotation is provided by an independent hydraulically reversible motor.

The need for hydraulic drilling arose because of a number of factors:

(i) A requirement for higher capacity drilling equipment to enable advantage to be taken of the increased loading rates being achieved with the LHD type of equipment;

(ii) A need to improve the working environment; and

(iii) A need to improve productivity per man due to an acute shortage of experienced miners.

The tunnel contracting industry in Europe and the metal mines in Scandinavia were the first to use and appreciate the advantages of the equipment. The first drill was introduced into Canada in 1970 and into the U.S.A. in 1973. Serious application of the concept in Canadian mines began in 1974.

The latest figures available (1979) indicate that there were approximately 1500 hydraulic drills operating throughout the world, of which 100 were in the U.S.A. and over 100 in Canada. There are now about 16 companies producing hydraulic drilling equipment.

There is no doubt, from the response to the questionnaire and from the plethora of technical papers being published on the subject, that the application of electric hydraulic drilling will increase rapidly in metal mines.

Hydraulic drilling has many attractions and a few disadvantages when compared with pneumatic drills. These can be described as follows:

Advantages

(i) Increased Penetration Rates and Productivity

In almost every instance where hydraulic drilling has replaced pneumatic drilling, the resultant penetration rate has doubled. As an illustration, a 20' x 14' heading was regularly drilled with 52 x 14' deep 2" diameter holes in 2½ hours by hydraulic drills, compared with 5 hours by pneumatic drills. Penetration rates of 4 fpm were obtained in conglomerate with button bits.

(ii) Better Environmental Conditions

(a) Visibility

There is a complete elimination of the fog associated with pneumatic drilling. This improves visibility and the working environment. In addition, the elimination of fog coupled with the availability of electric power, enables good lighting to be provided, thus creating a further improvement in visibility and working conditions.

(b) Noise

There is a considerable difference in the sound levels produced by hydraulic and pneumatic drills. This is because air exhaust noise is eliminated, and oil has a dampening effect on the piston blow noise. Peak sound levels are higher for hydraulics (3000 - 8000 Hz vs 100 - 1000 Hz) but this level is attenuated better by ear protection equipment. It should be noted, though, that this level is more damaging so protection equipment is more essential.

The hydraulic drill is therefore quieter and easier to attenuate using personal protection equipment. Sound levels have been reduced from the 120 dBa level with pneumatic drills to the 100 dBa level for hydraulic drills. Quite a considerable improvement.

(iii) Operating Costs

It is gradually becoming apparent that as operating and maintenance techniques are established, and as labour and management become skilled in the use of the drills, operating costs are becoming much lower than with pneumatic drilling, for a variety of reasons:

- Greater productivity per man - typical improvements in various mines show the number of holes drilled per crew increasing by over 100%, and tonnages per hole due to greater hole spacings and burdens increasing by 20%.

- Requires less power, since electric hydraulics are more efficient than pneumatics - for example power costs per foot drilled were 0.40 for hydraulic and 1.12 c for pneumatic drills made by the same manufacturer.
- Maintenance costs can be up to 50% less than for pneumatic. This is because the hydraulic drill has fewer moving parts, produces less vibration and is fully lubricated at all times.
- Drill steel and bit lives have increased considerably. This is mainly due to the more efficient transfer of energy by the smaller diameter pistons in the hydraulic drill which lowers the level of the stress waves in the piston and steel. Savings of up to 56% have been achieved with bit lives having increased at one mine from 40 m (pneumatic) to 190 m (hydraulic).
- The ability of the hydraulic drill to drill larger diameter holes (44 mm versus 38 mm) with no loss of penetration rate often means that fewer holes per round are required, and savings in drilling and charging time can be made, thus reducing costs further.

One mine reported saving $146,000 in one year by using hydraulic drills instead of pneumatic drills. Overall, savings in operating costs seem to range from 25 to 50%.

(iv) Versatility and Control

Hydraulic energy is easier to measure and control since the flow, pressure, stroke, thrust and rotation can all be adjusted so that penetration rates, rod and bit lives can be optimized for maximum overall efficiency in each application.

The ease of control enables automation to be readily applied to the hydraulic drill to provide such features as automatic stop and retract, automatic collaring and feed control, automatic reversal of the drill at mid-hole, and feed position indicators. These all have the additional advantage of reducing the operator's work load, enabling him to more effectively operate a multiboom rig with obvious benefits to productivity and miner morale.

It is also but a single step to the fully automatic remotely controlled drilling unit - a system of very recent design and operation described later in this section.

Disadvantages

(i) Capital Cost

The major disadvantage, especially in these days of high interest rates, is the capital cost. The capital cost of a hydraulic drill is approximately twice that of the pneumatic drill. This high initial capital cost is, however, more than offset by the benefits accruing from higher production outputs and lower operating costs.

(ii) Maintenance Labour

Hydraulic drilling equipment requires maintenance labour that is more skilled and conscientious than does pneumatic equipment. As personnel are trained and become skilled in the operation and maintenance (particularly planned maintenance) of the equipment, this problem will reduce.

Table 4-1 summarizes these comparative advantages and disadvantages of the hydraulic and pneumatic drilling systems. These are shown as ratios to make it easier to compare data based on various time bases.

Application

Electric hydraulic drilling is now being applied in underground mining in the following areas:

- Jumbo drilling for development headings, access drifts and stope drilling for room and pillar mining.

- Rig drilling for fan, ring and longhole drilling in sublevel caving, sublevel stoping and block caving.

The higher capital cost, and the greater capabilities and productivity do, however, demand that the fullest use be made of the equipment.

Not only does this mean that management has to lay out the mine to provide sufficient working faces to fully utilize the drilling equipment, but since the drilling operation is only one segment of the mining cycle, it is necessary to match the capabilities of the charging, loading, haulage and ground support equipment to the increased drilling potential. One item of equipment cannot be viewed in isolation when it forms part of a cycle of operations.

In development and access drivages, the higher drill rates and resulting reduced cycle times have enabled larger roadways to be driven at faster overall rates, while the goal of a full cycle in a shift is still attained. Larger roadways in ore permit larger loading and haulage units to be used, and more ventilation supplied at lower resistance and hence lower cost.

At the other end of the scale, the recent introduction of the single boom low profile drilling unit has enabled the benefits to be applied to roadways down to 5 m^2 section. The potential thus exists, particularly where the new high capacity loading and haulage systems are also applied, to greatly improve the rate of drivage of the headings driven in rock (usually kept as small as possible for secondary drivages).

The ability to improve drivage rates in development and access drivages reduces the amount of time required to (a) bring a mine into production and (b) develop new areas while providing more flexibility to management to meet the inevitable problems associated with varying ore grades.

An illustration of the potential rate of drivage is provided by a recent tunnel project. A 3 m horseshoe tunnel was driven in granite using an electrohydraulic twin boom rail jumbo, Haggloader and Shuttle train. The maximum rate achieved was 107 m in a 16 shift week with an average over 4 months of over 70 m/week.

Room and Pillar mining provides the ideal 'mining' application for hydraulic drilling because the layout enables several working faces to be available within a small area. It must be remembered that moving the drill about the mine, setting up, repositioning booms etc. takes just the same time with the hydraulic drill as with the pneumatic drill so that the greatest benefits of hydraulic drilling are obtained when it is actually drilling.

A typical room and pillar operation will probably require between 9 and 11 faces per equipment set to ensure that there is always a face available for drilling.

As an example of the relationship between the various equipment segments of the cycle, and the 'ultimate' in production from room and pillar systems, it is interesting to look at the proposals for oil shale mining in W. Colorado, U.S.A.

One company proposes to mine 60' high x 60' wide in two 30' lifts leaving 60' x 60' pillars. The 30' high top face will be drilled with a special two-boom hydraulic drill jumbo drilling 18 - 4½" dia. holes - 30' deep, to pull 28' and produce 3,300 tons per blast. (Powder ratio 0.5 lb ANFO/ton.)

There are nine faces required per panel operated with:

1 - Drill Jumbo;

1 - ANFO charging unit;
3 - 11 yd^3 loaders (FEL);
9-12 - 85 Ton Trucks; and
2 - Rock bolting machines.

The drill can drill three headings per shift, the loader muck one heading per shift, for an output of 68,000 tons per day.

The application of hydraulic drilling to cut and fill mining is difficult to justify since the horizontal drilling method reduces the available face room, vertical drilling, however, makes more working face area available.

The application of hydraulic drilling equipment to ring or fan longhole drilling follows as a logical extension to the jumbo applications, although again, the main criterion is to have sufficient drilling available to keep the machine busy. This equipment is now well into the development stage and is being adapted and used for cable bolting and cable and drainage holes in addition to its normal longhole duties in sublevel caving, sublevel stoping and block caving systems.

Examples of the various applications of hydraulic drilling will be found in Part 5.

Automation and Optimization of Drilling

Since the first mechanized drilling rigs were introduced in the 1960's, attempts have been made to automate the drilling functions and optimize the drilling parameters.

Pneumatics do not readily lend themselves to these functions, and automatic operations have been limited to:

- automatic stop and return;
- automatic flushing;
- parallel holding of tunnel booms;
- automatic uncoupling of longhole drill rods; and
- constant feed pressure system.

This is because pneumatics are difficult to monitor and regulate, and because they react slowly due to the compressibility of air.

In order to automate a rock drilling system, the system must be accurately monitored and the controls must be accurate and fast. However, to optimize the drilling parameters, events must be measured accurately and quickly, and fine and rapid adjustments to the relevant functions must be made. All these parameters are readily dealt with in the hydraulic system.

In addition to the features listed above, the following automatic controls are now possible with hydraulic drills:

- automatic anti-jamming (automatic feed reversal and re-engage);
- automatic collaring;
- automatic drill rotation reversal at mid depth. This increases steel and bit lives by up to 50%;
- automatic bit changing;
- automatic boom positioning; and
- automatic steel changing.

Many of these refinements are expensive, but as skilled labour becomes more expensive and scarcer, and as safety and control emphases increase, many of these features will become standard on future rigs.

With any high cost machine, it is imperative to obtain the maximum efficient use of the machine. This does not necessarily mean necessarily the highest penetration rate.

.2 In-the-Hole (ITH) Drills

The concept of in-the-hole drilling (or down-the-hole (DTH) as it was originally known) involves the use of a percussive drill which goes down the hole as it drills. It is pneumatically powered with air supplied through the drill rods. Rotation is supplied at the top of the drill stem. The piston strikes directly on the shank end of the bit with no power lost in transmission of the blow energy in the drill stem. There is thus almost no loss of penetration with hole depth.

Most current machines now use boosted air at 14 - 20 bar, with consequent improvements in performance (Fig. 4-3). Continuous hole clearing is maintained by the exhaust air from the drill. Bailing velocities in the hole must not be less than 16 m/s. Rotation and all other operations except the ITH Drill are now available hydraulically, powered by electric motors. The most common hole diameter is 165 mm although drills are available for smaller (108 mm) diameters.

In-the-hole drills have been used for many years by the oil industry, usually driven by the drilling mud, but their use in underground mining stemmed from the need to improve the effectiveness of the longhole (51 mm dia.) system which was in widespread use, but which suffered from severe limitiations in directional control, length of hole drillable and hole closure in poor ground.

The use of ITH drills has had a dramatic effect on mining operations and mine layout; some major advantages and disadvantages are highlighted below.

(i) Advantages

- Increased safety - all work from supported ground
- Higher penetration rates, especially with H.P. air and button bits
- Reduced noise levels (81 dBA v 117 dBA)
- Reduced dust and 'fog'
- Greater productivity (40% - 100% increase) stope drilling times reduced by 50%
- Longer holes feasible (up to 60 m) hence less development required as sublevels further apart (15 m v 55 m)
- More accurate drilling (1% deviation at 60 m) hence better blasting results.
- Larger diameter holes allow easier charging and use of spherical charges. Large efficient blast possible
- Able to drill in fractured ground. Hole closure no major problem
- Low maintenance and good availability
- Low costs per tonne, though costs per metre drilled are higher than 51 mm dia. longhole drilling.
- Quicker and easier to manoeuvre, move and set up. Physical size ideal for underground operations.
- Less physical labour involved and better working environment
- Better powder factors, better fragmentation, larger blasts
- Permits improved methods such as drop raising to be used for slots
- Able to drill up, down and slant holes
- Has enabled VCR to be introduced with consequent benefits to safety production and working environment.

(ii) Disadvantages

- Considerable care required in initial alignment to ensure benefits in blasting
- High pressure air through booster compressor required for optimum performance
- Bit type critical for specific ground conditions
- Greater operator skill required, as alignment is more critical.

Applications

In-the-hole drills are used for both up, down and slant drilling in sublevel stoping, large diameter blast-hole stoping, vertical crater retreat stoping, inverted benching, and any other mining method where large diameter holes can be effectively used.

The units are also finding increasing application for drilling for cable and drainage holes, ventilation holes for conventional or

Alimak raises, but more particularly for the formation of slot raises by the drop raise method utilizing the explosive cratering method.

The success of the DTH drill in the formation of slot raises is probably the main reason why the raise borer has not been so widely used in Canadian mines compared with metal mines overseas.

Generally it can be said that the introduction of the ITH drill has enabled open-pit mining techniques to be introduced underground, and has permitted the successful application of the spherical crater charge. These have led to the development of the Vertical Crater Retreat (VCR) and large diameter blasthole methods of mining with the consequent increases in output per drilled metre ranging from 6 to 10 times previous levels. However, it must also be borne in mind that drilling is only part of the whole operation and developments in loading techniques and ground support (described in sections 4.2 & 4.4) have also had their part to play.

Examples of the use of ITH drills are given in Section 5.

Another use to which the ITH drill is being applied is the recovery of pillars, where its ability to drill in broken or stressed ground is invaluable.

.3 Computer Controlled Hard Rock Drilling

Although the developments mentioned previously have resulted in a high level of mechanization of the drilling process, there are still a number of shortcomings in practical mining and tunnelling operating situations.

(i) Although detailed drilling & blasting plans are worked out, operators often tend to use their own patterns which they believe are the best; these frequently vary from gang to gang and from the official plan.

(ii) Contour holes are often randomly placed with consequent effects on rock stability.

(iii) On multiboom jumbos, one boom is often idle while the operator attends to some other problem. The time spent moving the boom from one hole location to the next depends greatly on the skill of the operator, and hence the total drilling time can vary considerably.

(iv) Penetration rate of the drill hammers is dependent on the adjustment of several parameters - percussive energy, feed pressure, rotational speed. Optimum conditions vary in each hole and from hole to hole. Hammer parameters should be constantly measured and modified to optimize performance.

The problems were first investigated in 1972 and whilst full automation was found to be technically feasible, the cost was high and the market was not receptive.

Work restarted in 1977 and a three boom jumbo produced and tested in 1978. The Norwegian company's experience now includes over 500 000 m of 45 mm dia. holes drilled by microprocessor controlled jumbos. The latest versions are equipped and programmed for the automatic and precise drilling of complete rounds according to predetermined drilling plans and fixed coordinates.

The use of this sophisticated equipment has become justified by the loss of skilled labour, increases in the cost of labour and the availability of cheaper more reliable computer control systems.

The advantages of the equipment are:

- Improved operational performance & drilling precision;
- Lower maintenance due to smoother action of computer control;
- More control over roadway profile and better ground conditions & stability;
- Record maintained of all parameters, hence easier to make improvements; and
- Improved operator workload.

The possible disadvantages are:

- Higher capital cost; and
- Greater maintenance skills required, especially electronically.

The details of the system and its operation are given in Part 5.

Applications

To date, the computer controlled drill has only been applied to underground civil construction works. However, like many developments initiated by the construction industry, the equipment should have a future application in the mining industry which is plagued by the same problems of lack of skilled labour and rising costs. Although the computer control has only been applied to the drill jumbo for tunnel excavation, there appears to be no reason why it should not be applied to other drilling operations underground, such as fan & ring drilling, where accuracy of direction and length of hole are critical.

.4 Trends

Electric-Hydraulic Drilling

There is no doubt that the use of electric-hydraulic equipment is becoming more widespread in underground metal mines, though Canadian mines have been slower than their international counterparts to accept the equipment.

However, responses to the questionnaire left no doubt that this is the major trend in standard drilling equipment for the future, because it offers that leap forward in productivity needed to combat rising costs, falling grades and manpower shortages, and offers considerable cost savings, reductions in energy requirements and improvements in environmental conditions.

Hydraulic drilling will be the favored tool in the future, when large footages of drilling are required in both development and production operations.

The development of rigs specifically for each type of mining system has started and will continue; however, the next forseeable development will be the application of airleg mounted hydraulic drills and stopers for use in small headings and cut and fill or open stoping. Hand held hydraulic drills and breakers are already in regular use in the construction industry, an industry which has been the forerunner of many mining developments.

There is therefore considerable potential for the conversion of Jackleg and Stoper hand held equipment from pneumatic to hydraulic operation.

The introduction of hydraulic drilling has enabled the automation of drilling functions, and the optimization of drilling parameters to be developed, leading to the production and operation of the first fully automated computer controlled drilling rig.

There is now little doubt that hydraulic drills will eventually take over the entire market.

ITH Drills

ITH drills will find greater application in underground mining where ore deposits are mineable by VCR, sublevel caving and large diameter blasthole stoping. The ability to blast ore in horizontal slices across the complete stope from the bottom up by the cratering principle, and to store the ore in the stope to support the walls could lead to sublevel stoping being applied to ore bodies that can only be mined presently by the shrinkage stoping method.

In terms of the actual machinery, the use of hydraulics electrically powered for rod rotation and other movements excluding the ITH drill unit should increase because of the greater efficiencies available with hydraulics.

It is unlikely that the use of hydraulics for operating the ITH drill will be considered because of the problems of supply and recovery through the drill rods, and the removal of cuttings. However, if water flushing were acceptable, the use of invert emulsion water/oil mixtures (5% oil, 95% water) may yet become common.

The ability of the ITH drill to be used in all attitudes could lead to its use for other longhole operations, such as ring and and fan drilling, in ore bodies where the spherical charges and greater burdens can be utilized.

The continued development of drilling techniques and drill bits should lead to the ability to drill longer holes (up to 125 m) which will reduce further development time and costs and greatly improve production efficiency.

In this develoment is the need to improve methods of aligning DTH drills and of further reducing borehole deviations.

Other possible applications are for bench drilling in multi-lift room and pillar mining.

Computer Control

The present application of computer controls is only the beginning of the application of sophisticated techniques. The trend, following the initial introduction, should be the application of this technique to other forms of drilling underground, followed by:

- automatic drilling without supervision;
- optimization of drilling parameters, particularly hammer work;
- automatic monitoring and warning of equipment failure; and
- application to support operations such as rock bolting, shotcreting and scaling.

It is likely, however, that the automation process will be next applied to semi stationary units such as raise borers, DTH drills and diamond drills.

4.3 LOADING

431 HISTORICAL DEVELOPMENT

The earliest forms of loading in mines utilized either gravity or hand mucking. By the 1910's pneumatic scrapers or slushers and rocker shovels were appearing, to be followed by electric scrapers, loaders & rocker shovels by the early 1920's.

The rocker or overshot shovel loader led to the hopper loader, an overshot loader with a hopper taking 6-10 passes of the bucket. Once full, the machine transferred to a central discharge point close by. The hopper loader in turn can be classed as the forerunner of the Load Haul Dump (LHD) concept. Initially pneumatic, but mainly diesel powered, these trackless vehicles were first introduced in mid 60's into Canadian mines. The LHD unit, as its name implies, has a bucket which is loaded in one pass and hauled to a central dumping point. Front-end loaders in larger mines served the same purpose for part of the cycle, working in conjunction with some other form of haulage, e.g. truck, rail car or conveyor.

One type of loader, first introduced in 1917, was the gathering arm loader. This loader was never widely used in metal mines (except in Sweden) because of its inability to handle abrasive material. However, the attraction of continuous loading has led to the development of the gathering arm principle in a form suitable for handling hard rock - the Haggloader introduced in 1972.

432 USES OF LOADING EQUIPMENT UNDERGROUND

The major loading principles and equipment types in use in underground metal mines can be summarized as follows:

.1 Development and Access Drivages

(i) Overshot or rocker loaders - usually rail or crawler mounted and used in smaller headings 5-20 m^2 section - pneumatic or electric powered - (Level to 1 in 4). (Fig. 4-4)

(ii) Gathering Arm Loaders (Haggloader) - high capacity continuous loaders used in headings from 5 m^2 upwards in section - pneumatic, diesel or electric (level to 1 in 6). (Fig. 4-5)

(iii) Load/Haul/Dump Units - rubber tired with bucket capacities ranging from 0.5 m^3 to 11 m^3 - used in headings from 4 m^2 upwards, powered by pneumatic, diesel and electric units (level to 1 in 7 production, level to 1 in 4 drivage). (Fig. 4-6)

(iv) Front-end Loaders (FEL) - rubber tired diesel and only used in the largest of excavations where space to maneouver to discharge into trucks is available.

(v) Scrapers · electric or pneumatic used in steep drifts, exceeding 1 in 4. Use in normal headings now unlikely.

.2 Production Stopes

(i) Gravity - almost solely confined to block caving and square set stoping.

(ii) Scrapers - still widely used in the electric and occasionally pneumatic form over distances from 15 to 125 m in shrinkage, sublevel and cut and fill stoping, block caving and square set stoping.

(iii) Load/Haul/Dump - probably the most widely used and versatile form of loading. Over 6000 in use in Canada and USA for VCR, blasthole, shrinkage, open, sublevel, and cut and fill stoping, sublevel and block caving and room and pillar mining. Available in pneumatic, diesel (majority) & electric forms with a wide range of bucket capacities. Performance and efficiency drop drastically with increasing distance from draw-point to dump, 1 m^3 machine over 100 m, 8 m^3 machine over 1000 m.

(iv) Front-end Loaders - usually diesel powered, rubber tired, with bucket capacities up to 18 m^3, these units are most widely found in large scale room and pillar mining, and for draw point loading in other forms where LHD operation are not considered to be practical or economical.

(v) Gathering Arm Loaders - this type of loader is used widely in room and pillar mining and also in certain countries in sub-level caving operations, where advantage is taken of the high loading capacities (120-180 m^3/h).

433 RECENT DEVELOPMENTS AND TRENDS

The main areas where developments have been taking place in the past few years are:

LHD - Electric powered through trailing cable
- Electric powered from overhead trolley wires
- Remote Control

G.A. Loaders - Hagglund developments

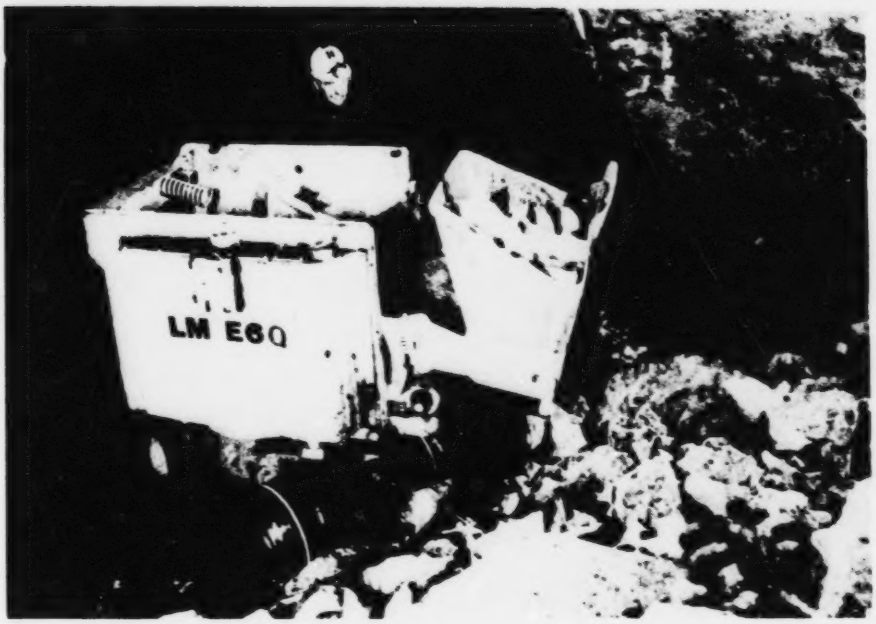


Fig. 4-4
Electric hydraulic
rocker shovel
(Courtesy Atlas Copco)

Fig. 4-5
Haggloader gathering arm loader filling
shuttle train in small development
(Courtesy Atlas Copco)

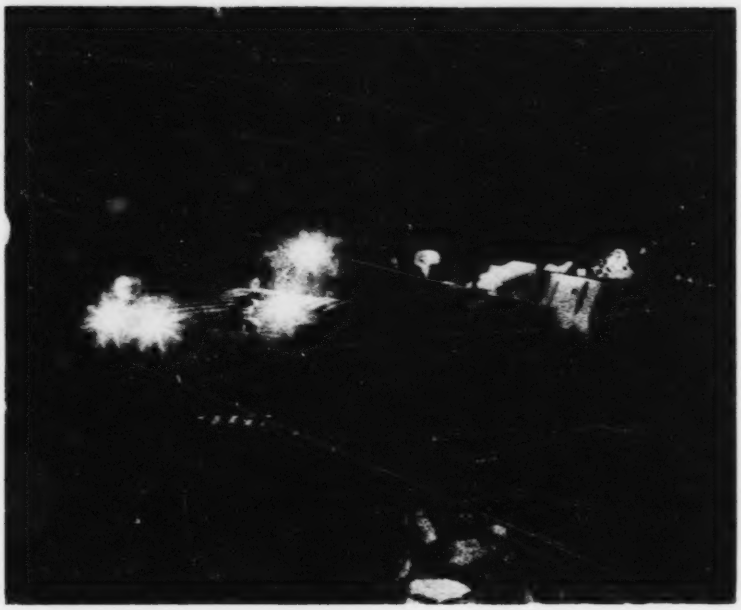


Fig. 4-6
Low profile Load Haul Dump (LHD) unit
in production stope
(Courtesy Toro)

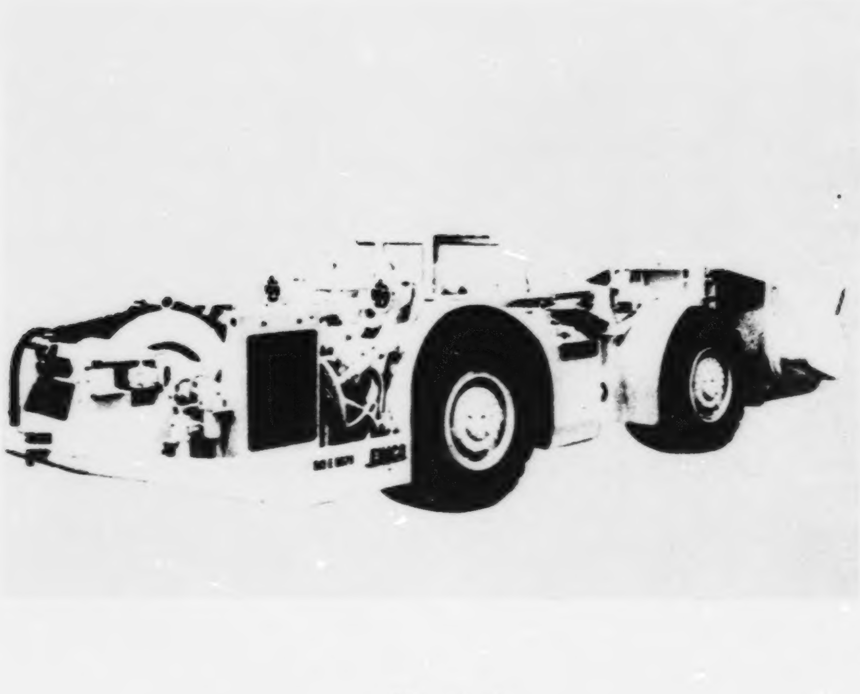


Fig. 4-7
Electric L.H.D.
(Trailing Cable)
(Courtesy Eimco)

.1 Electric LHD (Trailing Cable)

The introduction of the diesel LHD in the early 1960's led to a vast explosion in the use of trackless mining, but brought with it initially a need for larger roadways and increased ventilation to dilute diesel exhaust gases (oxides of nitrogen and carbon monoxide) to the levels required by the mining regulations. At that time, diesel fuel was cheap, and energy costs were low, and the resulting reduction in the rate of cost increase, and the increase in productivity were more than welcome. The LHD provided greater haulage speeds, higher maneouverability, and lower maintenance and labour costs per tonne.

Following the early designs, it became apparent that the LHD had to become narrower and lower in height to work in many ore bodies, leading to the development of low profile equipment.

In most Canadian mines, due to the harsh winter climate, it is necessary to heat the intake air. The combination of high diesel fuel cost, high heating fuel cost, and the need for greater dilution of the diesel exhaust fumes to provide adequate ventilation is now leading to a rapid re-appraisal of the diesel LHD and a hard look at the electric LHD.

The first prototype unit was developed in 1972/73 equipped with a trailing cable mounted on a installed cable reel, and used hydrostatic components. Modern versions now utilize DC drive motors with solid state control components. Cables & reels up to 125 m capacity are now available. (Fig. 4-7)

The electric LHD is gradually becoming more acceptable in metal mines. Its principal advantages and disadvantages are:

Advantages

- Lower ventilation requirement, hence lower ventilation cost in some mines. Ventilation cost (including heating) can be up to 34% of operating cost per tonne.
- Lower noise levels (90 dBA v 110dBA).
- Better visibility.
- Increased production (+15% to 20%) because of increased availability.
- Improved environmental conditions - lower operating temperatures (at least 50°F), less dust and particulate matter.
- Easier and smoother controls, faster responses, better operator acceptance.
- Lower tire costs (wheel spin restricted by current limits).
- Less routine maintenance, less wear & tear.
- Lower operating costs (25% less).
- More energy efficient (90% vs 45% for diesel).
- No auxiliary ventilation required.

- Lower fire risk with elimination of diesel fuel.
- Greater break-out forces possible due to overload characteristic of electric motors.

Disadvantages

- Range of operations limited by capacity of cable reel. (Usually 125 m maximum - permitting approximately 250 m travel with mid point supply.)
- Unless properly managed and cared for, cable can be easily damaged.
- Additional electrical staff required for maintenance.

Trends

As mines become deeper and environmentally hotter, the electric LHD will become a necessity, because of its cooler operating level and lesser ventilation requirements.

The soaring cost of diesel fuel and heating fuel will virtually cause mines to move to the electric L.H.D. (as part of a trend to the all electric mine).

There is also a trend - fostered by the high cost of fuel, and the efficiency losses inherent in a compromise machine (LHD) which is called upon to do two different functions, load and haul - to use loaders in the stope and a separate truck for ramp haulage or long distance haulage. This concept neatly absorbs the electric LHD by removing the limit on operating distances imposed by the cable.

The trend in the 60's and early 70's was to diesel equipment, that of the late 70's and 80's will be towards electrically powered equipment.

.2 Electric LHD (Trolley Wire)

This is a very recent development and is intended to remove some of the disadvantages of the cable reel LHD. During the past two years, trials have been carried out in a metal mine with an electric 4 m^3 LHD operating from a monorail trolley pick up system.

The system comprises a three phase AC trolley pickup with four conductors (three power, one ground) carried on an I-beam monorail. The system can incorporate switches to permit vehicles to pass, and with the use of the cable reel on the LHD allows more flexibility, permitting the electric vehicle to operate away from the trolley system:

The additional advantages of this system over the cable reel only system are:

(i) Increased flexibility and operating range; and

(ii) Ability to operate more than one vehicle on the same system without cable complications.

Possible disadvantages are:

(i) Potential danger from live overhead wires (dealt with by warning signs, lights and good training and safety procedures); and

(ii) Problems associated with roof maintenance (scaling).

The system is being extended to a truck haulage operation and has particular advantage for ramp and incline haulage.

There is no doubt that this extension of the electric LHD/truck will have great potential in underground metal mines.

The above system is described in detail in Part 5.

.3 LHD - Remote Control

Some operations in underground mining are more hazardous than others. In loading operations, these include drawpoint loading and cut & fill loading, where the operator or the vehicle is exposed to sudden movement of ore, or is working under an unstable back, respectively.

The ability to remotely operate the loader in these circumstances provides greater safety for the operator, greater productivity, permits stopes to be emptied thus enhancing ore recoveries, and provides more flexibility. In one instance, standard draw points in the VCR method have been eliminated with a consequent reduction in development and costs.

By 1980, there were over 20 radio controlled LHD's operating in Canada. (Fig. 4-8).

Remote control can be effected in one of two ways, either by umbilical cord or by radio control. The first systems used in Canada in 1975 were based on the umbilical cord. These were found to be effective but somewhat cumbersome for the operator and limiting in operating distances. The system does, however, have advantages in heading drivages with rocker shovels. However, they proved the feasibility of the system and encouraged the development of the radio controlled method in 1977. A portable console unit is carried by the operator, transmitting frequency modulated signals to a receiver on the machine. This unit allows the operator to control the machine at distances of up to 100 m. In normal operation, the LHD is operated by radio control at the drawpoint and within 30 m, but is manually operated to the ore pass and for dumping. The changeover is via a simple switch.

The radio control system is described in detail in Part 5.

The impacts of remote control have on mining operations can be summarized as follows:

- ability to increase recoverable reserves from stopes;
- increased revenue resulting from 100% stope recovery;
- reduced stoping costs;
- decreased development expenditures and development time;
- increased employee safety and operating conditions; and
- capital costs of unit ($20,000+) quickly recovered from increased ore recovery benefits. One mine estimates it recovers an extra 18,000 t of ore from each stope using remote control LHD's.

Future Applications (Trends)

There appears to be no reason why radio remote control should not be applied to any mining operation where there is some degree of risk to the operator.

Such operations as scaling, rock bolting, loading explosives and application of shotcrete would appear to be prime candidates for its application.

.4 Gathering Arm Loaders

The basic form of gathering arm loaders incorporated the gathering arms in the gathering head, which was buried in the ore during loading. This usually resulted in poor service lives and high maintenance costs when handling hard abrasive ore.

This type of loader is still in use in the Swedish iron ore mines, loading directly into trucks, where its high loading rate can be utilized fully. The loader is widely used in mineral mines including coal (i.e. non-abrasive conditions).

The development that has re-kindled interest in gathering arm loaders is the Haggloader. This utilizes 'overhand' gathering arms which gather the ore/rock onto the gathering head from a position above the ore. The machine, which has a high loading rate - 120 - 180 m^3/hr, is available with pneumatic or electric power sources (the latter being the most popular - 95%) and for use on rails, crawlers or rubber tires. (Fig. 4-9)

The major advantage of this type of loader is that it loads continuously and smoothly without throwing or jerking the rock. It can discharge into a shuttle train (railborne) or into trucks or similar vehicles, or via bridge conveyors onto a belt conveyor system.

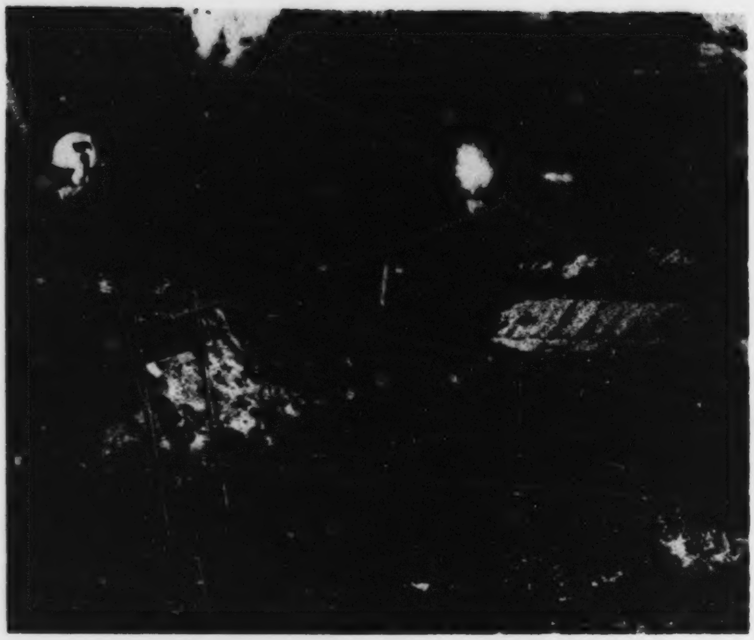


Fig. 4-8
Diesel powered L.H.D.
with radio remote control
(Courtesy Toro)

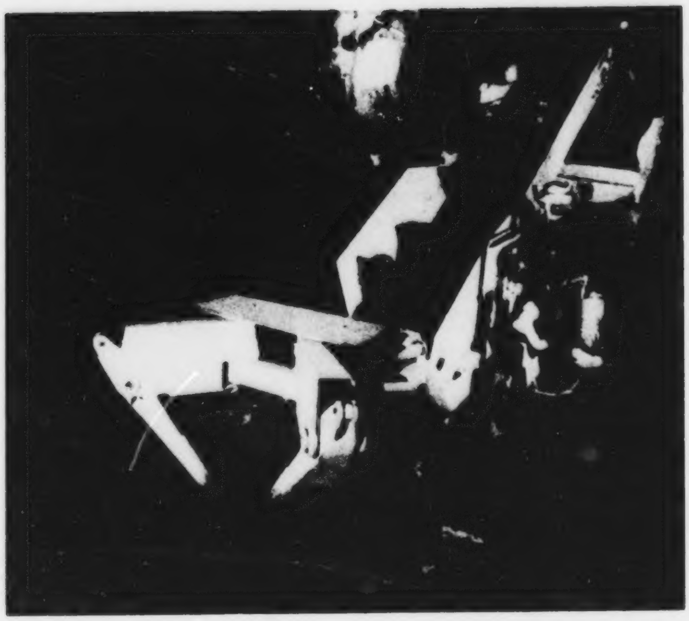
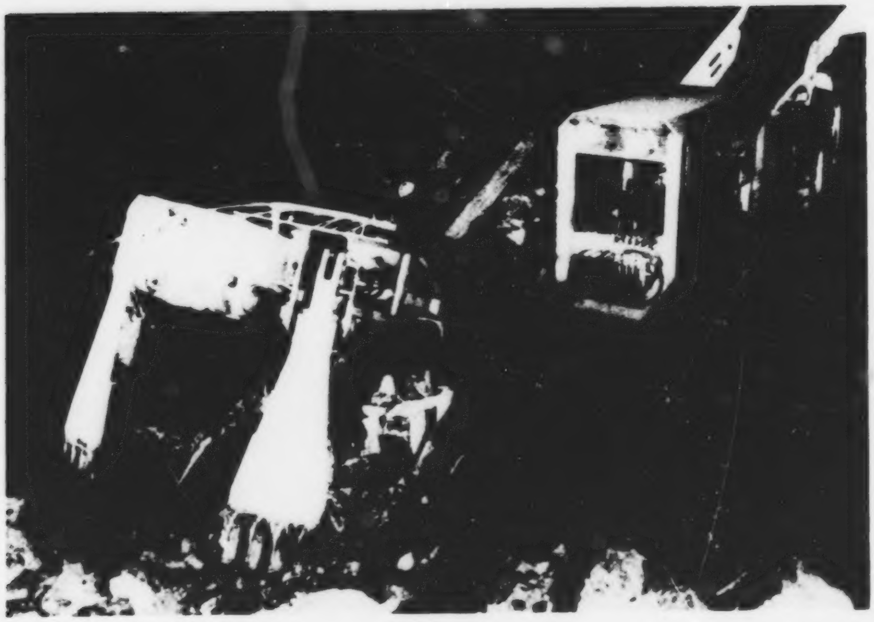


Fig. 4-9
Haggloader type gathering
arm loader
(Courtesy Atlas Copco)



Applications

The major applications for the Haggloader to date have been in tunnel or access drivages. Here the high capacity loader has been allied with electric hydraulic drill jumbos, and shuttle trains or trucks to develop high speed drill and blast drivage operations. These combinations offer a system capable of drilling 'tunnels' at rates between 1.3 and 1.8 times the rates attained with normal equipment.

eg. 10 m^2 section tunnel - 56m/wk v 34m/wk
8 m^2 section tunnel - 72m/wk v 45m/wk

Mucking out times are usually between 1/2 and 1/3 those using conventional equipment.

An area in which the Haggloader has considerable potential, but has yet to be applied, is in production loading from stopes. The machine offers high loading rates, continuous loading and a good operator environment. Allied to truck, conveyor or rail haulage, high production rates from stopes can be achieved.

The application of the Haggloader with the Haggland Shuttle train in high speed tunnel drivage is described in detail in Part 5.

There are currently seven Haggloaders being used in Canadian metal mines.

Advantages

- faster mucking, improved production (+40%)
- safer, smoother operation
- better visibility
- better operator environment
- less complex than LHD, maintenance less
- mine development time shorter from higher drivage rates hence financing costs less.

Disadvantages

- higher capital cost (3 x Rocker Shovel; Similar to LHD)
- limited to 30 m radius curve operating, 15 m radius travelling
- more complex than Rocker Shovel.

4.4 TRANSPORTATION

441 HISTORICAL REVIEW

Primitive methods (still in use in some third world countries) involved carrying minerals in sacks, skins or baskets. This developed via the wheelbarrow to the hand trammed mine car or tub. Mechanization of haulage started with the horse and pony; pneumatic and later electric rope haulages; and pneumatic, and later diesel, battery and trolley locomotives.

Conveyors form an important part of many haulage systems, having been introduced near the turn of the century in the form of shaker conveyors, and developing through scraper and chain to belt conveyors. Conveyors are no longer restricted to straight roadways. Designs are now available for curved, undulating and vertical applications.

Pipeline transportation of materials has had, until recently, little application in underground mining with the exception of materials for filling stopes, usually by gravity. In the past 10 years, systems have been developed for hoisting minerals to the surface.

442 GENERAL

One of the most important considerations in the efficient operation of an underground mine is the haulage system. Three aspects are involved, the rapid and economical removal of the ore from the working face or of rock from development headings to the surface; the transport of supplies in the opposite direction; and the safe and rapid transport of men in and out of the mine.

We are not concerned, in this report, with the relative merits of the many and varied systems available in these categories, but with the developing trends.

We have listed below the main equipment types available in the three main areas, and shall deal in more detail with the developments in the sections that follow.

Transport of Ore and Waste Rock

- LHD/FEL
- Truck
- Locomotive and Mine Cars or Shuttle train
- Conveyor (with or without Feeder Breakers/Crushers)
- Pipeline

Transport of Materials

- Diesel Vehicles
- Rail (Locomotive)
- Monorail

Transport of Men

- Diesel vehicles
- Battery vehicles
- Rail - Locomotive
- Monorail
- Rope - Ski lift
- Conveyor

443 Trends and Developments

.1 Transport of Ore and Waste Rock

.11 Load Haul Dump/Front End Loaders These have been dealt with in Section 433 in detail. The trends are to electrification by cable reel and trolley wire.

.12 Truck There have been three recent developments for underground mines, and one for surface mines which could have underground applications in the future.

(i) Low Profile Equipment

The initial impetus for low profile equipment came in the mid-1960's with the LHD vehicle; however, experience has since shown that these vehicles have their limitations particularly related to economical haulage distances, and for hauling up gradients, for example in ramp mining.

The need for trucks for underground use is now being met with the four wheel drive articulated low profile diesel powered truck. For example, 30 t trucks with an overall height of 2.35 m and capable of operating in 5.1 m wide roadways including right angle bends, and 55 t truck 3.55 m in height are now available. (Fig. 4-10)

The availability of low profile high capacity trucks has widened the applications for truck haulage underground, without the need for larger roadways.

One new type of truck incorporates a reversible chain conveyor in the body base which permits it to load and discharge without the need for additional operating height. (Fig. 4-11)

(ii) Container Units

A Swedish company has recently introduced a system of containers for the transport of ore underground. The system comprises a haulage unit and two containers, one container is left at the face to be filled, whilst the second container is picked up and hauled to the dump station. Switching containers at the face takes less than 60 s, thus virtually eliminating the loading waiting time. The haulage capacity per unit is substantially increased, compared to standard rear dump trucks which must wait to be loaded.

A system comprising two 40 t containers has a productive capacity of approximately 500 t/h, dependent on haulage distance. This represented an increase in production of 20% over a standard truck system.

(iii) Electric Trucks

In a similar manner to the LHD conversion, the electrification of underground mine trucks is being implemented. The high cost of diesel fuel, of fuel for heating the extra ventilation air required for the diesels, and the availability of a practical trolley wire system for use underground have added impetus to this move. Unlike the two wire trolley assist used for trucks in surface mines, the underground system is based on a four wire supply, with the facility for the operation to be extended beyond the trolley system using the cable reel on the vehicle. Reel capacities for 300 m travel are now available, although the initial test truck will only have a 30 m cable. The trolley system has proved to be capable of operation on curves but has yet to be proved on decline operations.

The equipment used to date is identical to that described for the LHD in Part 5.

In view of the apparent unlikelihood of a major breakthrough in fuel economy, the application of electric power to mine trucks obviously has great potential. Allied with the cable reel system, the electric truck now has the same flexibility as diesel equipment.

The system could be used on the level or ramp, applied to LHD units or any other electric mobile equipment such as electric hydraulic drill rigs.

(iv) Automatic Truck Control

This system has to date only been operated in a surface mine, and involves the automatic operation of mine trucks on the normal cycle between the loading face and the dump station. The system

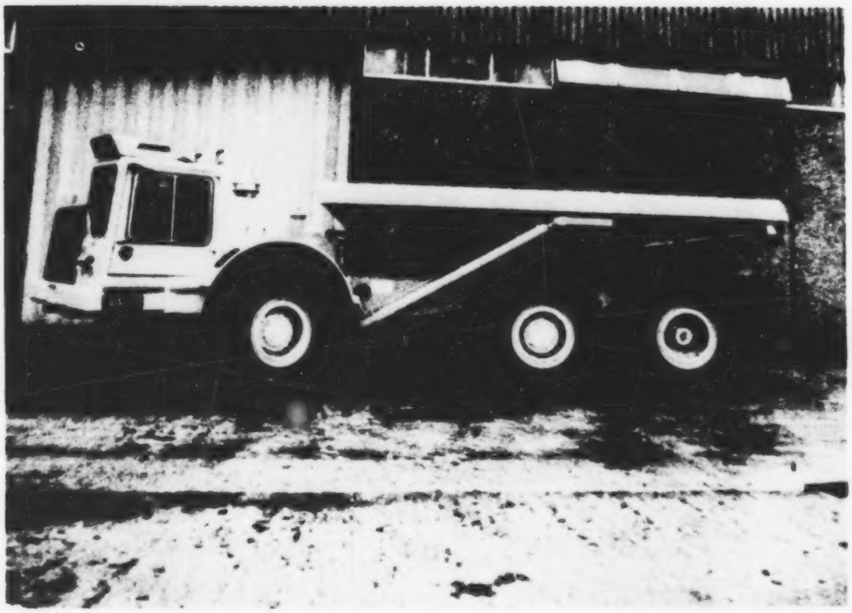


Fig. 4-10
Low profile articulated
haulage truck
(Courtesy DJB Engineering)

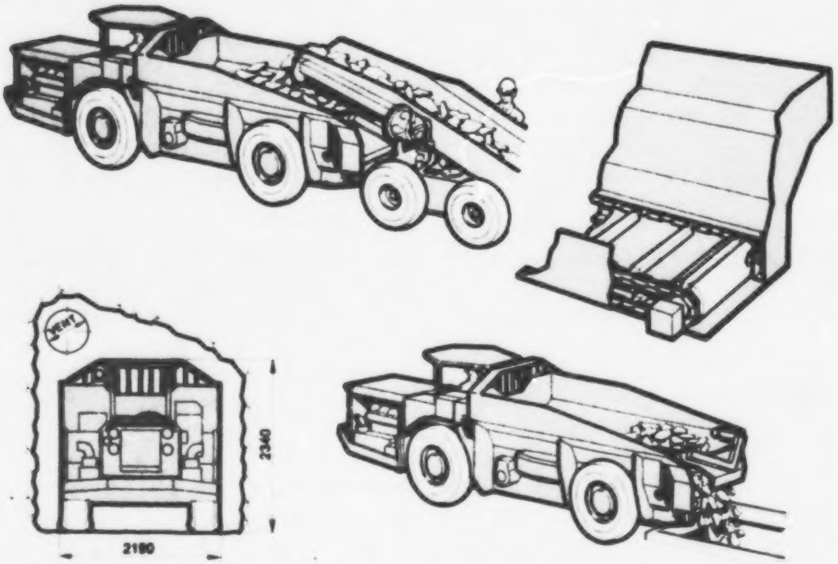


Fig. 4-11
Low profile truck
with chain conveyor
(Courtesy Atlas Copco)

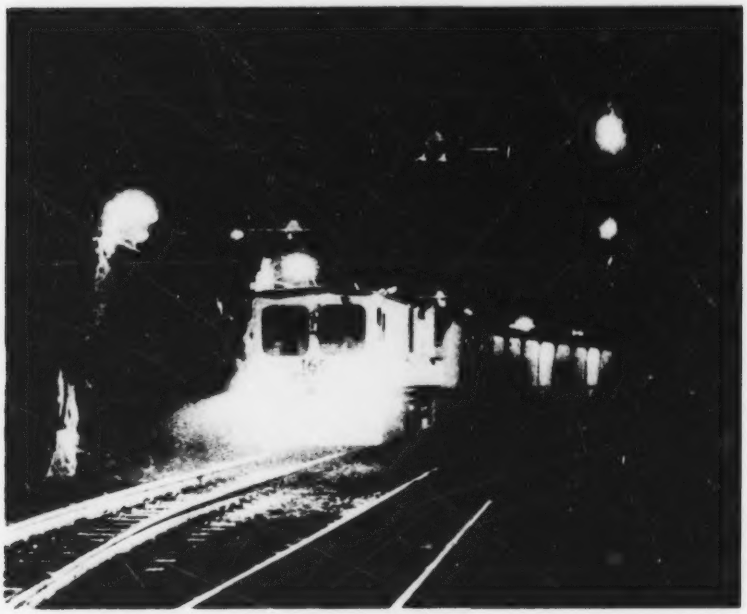


Fig. 4-12
Fully automatic trolley locomotive
ore handling system in Sweden
(Courtesy ASEA)

offers benefits of savings in labour, maintenance costs, lower fuel consumption for diesel powered vehicles, and safer operations. It is expected to be operational in the mines during the 1980's.

No assessment has yet been made as to its application underground, where operating clearances are tighter than on the surface, but viewed at this point in time, it does appear to warrant observing for future underground use.

.13 Locomotives and Mine Car or Shuttle Train

(i) Production and Development Haulage

Locomotives have generally been the mainstay for primary haulage underground. In Canadian mines, battery and trolley are in equal use, and there are a small number of diesel powered locomotives. (See Table 4-3.)

The major developments in recent years in the haulage of ore or rock are the introduction of remotely controlled and fully automatically operated trains, rack locomotives for use on steep gradients, trolley/battery units and shuttle trains.

(a) Remote and Automatic Train Operation

Technical developments in the field of mine locomotives, ore cars and control systems now make it possible to fully automate underground trackbound haulage for unmanned operation. Several such systems are now in service in mines all over the world, and they have proven that an investment in highly mechanized and automated equipment is beneficial for safety and working conditions as well as for productivity. (Fig. 4-12)

Statistics from all mines show that haulage operations have the highest accident and fatality rate. The introduction of automated haulage has the following benefits:

- Virtual elimination of personnel accidents
- Virtual elimination of collisions and derailments
- Increased utilization of rolling stock (1.8 times)
- Manpower requirements reduced (by 60%)
- Annual maintenance costs reduced (to 1/7th of manual system)
- Reduced production costs (45% of manual system)

A typical automated train haulage system will consist of electric trolley thyrister chopper controlled locomotives, mine cars, remotely operated loading stations, automatic unloading stations, and a centralized computer control centre from which the remote control system is operated. A typical system is described in detail in Part 5.

Increased effectiveness can also be achieved with less than complete automation. For example, trains can be operated manually over a rail system controlled by computer operated signalling, but with the loading and unloading sections of the cycle being undertaken by remote control.

Whatever the motive for automating underground rail haulage, the investment must be profitable. It is most profitable when applied to new haulage levels, since automating existing systems often requires rebuilding loading and unloading stations and replacement of rolling stock.

In making a choice between manually operated trains controlled by signals and fully automated driverless trains, it should be noted that the initial cost of automation is somewhat less than twice the cost of a manual operation with required signalling. Further, control equipment for complete automation typically represents between 10% and 20% of the cost of the new rolling stock.

With the shortage of skilled manpower remaining a chronic problem for the mining industry, largely due to the accident and fatality record, any system that can reduce these rates, improve the environmental and working conditions, reduce the need for manpower, improve productivity and reduce costs must be given serious consideration by the mining industry. The systems referred to have been in operation in the manual/signal mode for 20 years, but the automated system has been operating in mines for about six years, and has now proven its worth. As far as we are aware, only one Canadian mine has introduced a fully automated rail haulage system, in spite of the fact that rail haulage is the predominant transport method in use in Canadian mines.

(b) Rack and Adhesion Locomotives

One of the most significant developments in the past decade in the field of locomotive haulage has been the introduction of the rack locomotive for underground use. This unit incorporates a rack and pinion drive located between the tracks. Grades of up to 20% can be safely negotiated, but more importantly grades in the 8% - 15% range can be taken at normal locomotive speeds in complete safety without fear of a runaway. (Fig. 4-13)

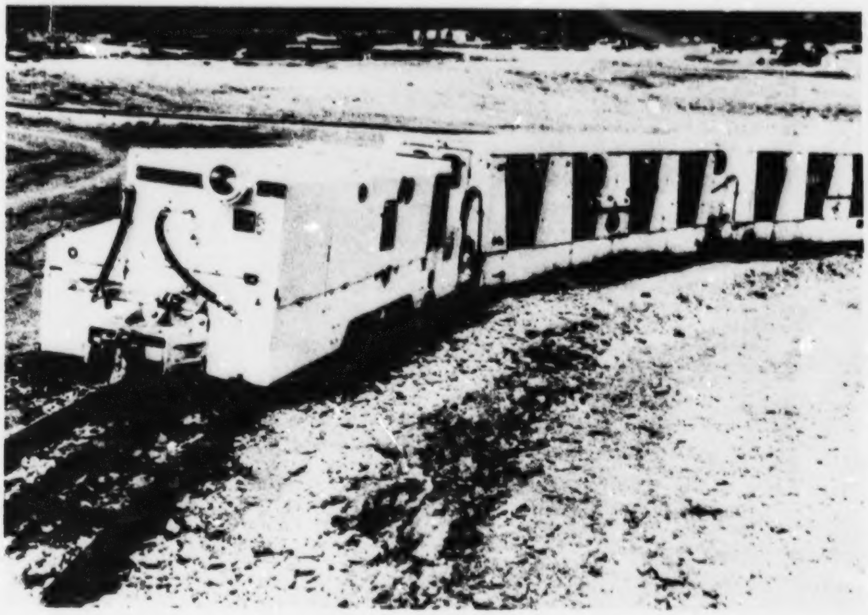
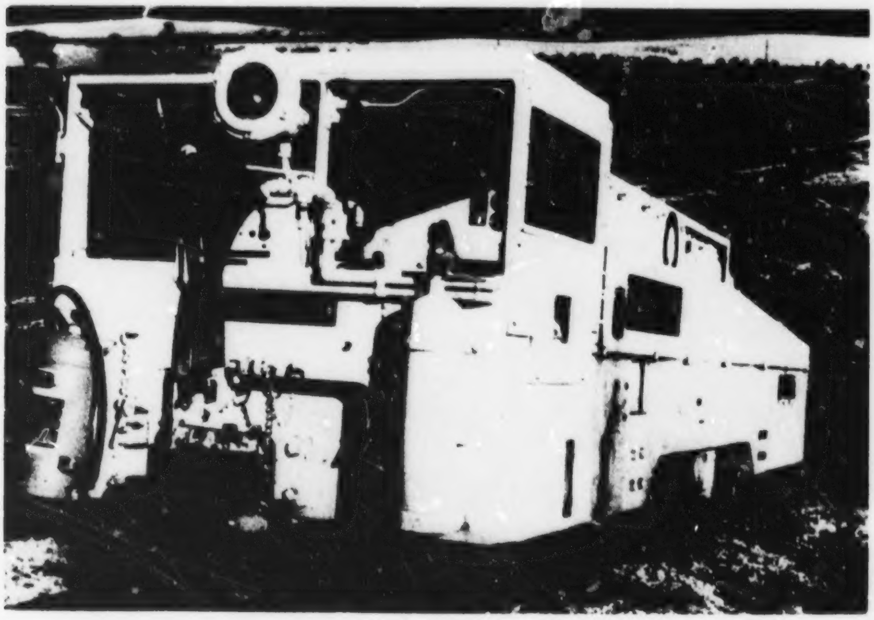


Fig. 4-13
Steep gradient rack drive locomotives
on 8% & 15% grades
(Courtesy - Hunslet
- GMT)



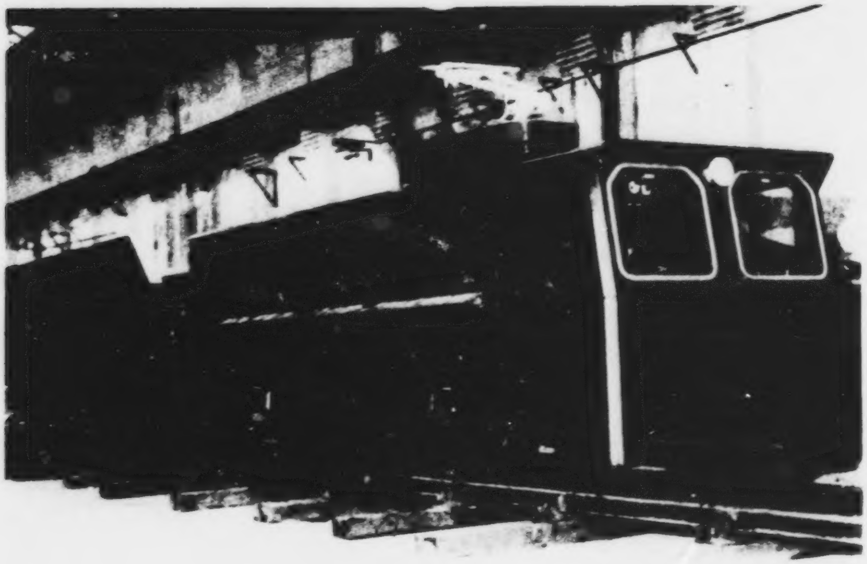


Fig. 4-14
Trolley-battery locomotive 25 T
in use at Mt. Isa
(Courtesy - Hudson
- Hunslet)

Fig. 4-15
Shuttle train transferring load
to next car in small heading
(Courtesy Atlas Copco)

A further development is the fitting of rubber tires onto the flanged steel wheels, giving greater adhesion to the rails. This development alone enables standard locomotives to be operated on gradients up to 9%.

Both systems can be used together to provide a reliable and safe means of handling mine cars, materials and personnel on steep gradients. These units (up to 160 Hp) have to date only been used in the U.K., but offer the potential to widen the scope of locomotive applications in mining.

(c) Battery/Trolley Locomotives

The latest of non-diesel locomotives being made available to the mining industry is a combination trolley/battery unit. This incorporates the facility for the locomotive to operate as a trolley unit on the main haulage level, and to operate on secondary levels away from the trolley wire system under battery power. The locomotive incorporates a charging unit which keeps the battery constantly charged up whilst the locomotive is connected to the trolley wire, thus making available the full battery capacity wherever it is operated on the secondary system. The development of solid state controllers for battery and trolley operation have also resulted in better traction, longer battery life, the elimination of contactor maintenance, finer speed control and regenerative braking.

These units provide considerable flexibility and the means to extend mechanized transport to all levels of the mine without the need for the extra ventilation required for diesel equipment (rail or trackless). (Fig. 4-14)

(d) Shuttle Trains

This is a concept which has been used for a considerable number of years in the tunnelling industry, but which has only very recently been applied in underground mining. Its prime application is in development drivages with rail mounted equipment. The shuttle train consists of a series of units. Each unit comprises a long narrow rail mounted body with a chain conveyor in the bottom, which facilitates the transfer of rock from one car to the next. The discharge end of each car is adapted to fit the loading end of the car behind (Fig. 4-15). A train usually consists of sufficient units to accommodate a full round, especially in small headings. The units are usually electrically operated, though pneumatic operation is possible, and are hauled from face to dump by locomotive. They are self-discharging at the dump.

The shuttle train, used in conjunction with high capacity loading equipment (e.g., Haggloader) and high capacity drilling equipment (electro-hydraulic) provides a system ideally suited to high speed drivages of mine roadways. Such a combination has been used to drive a 9.7 m^2 section tunnel through sandstones of up to 2,500 kg/cm^2 compressive strength. In the last two months of the drivage (which was completed 18 weeks ahead of schedule), the time for drilling, blasting and mucking out was approximately 3 h. The potential for reducing development costs and time in underground mines with this equipment is considerable, as indeed are the benefits.

The project utilizing the above equipment is described in Part 5.

.14 Conveyors Belt conveyors provide a means of continuously transporting ore or rock over short or long distances underground; however, for belt conveyors to be cost effective and practical for underground metal mines, it is always necessary to crush the ore to a size suitable for conveying. Dependent on the size range, the limits usually applied are that lump size should not exceed 40 - 50% of the belt width.

Crushers for this purpose can be of two types, either a stationary primary crusher to which the ore is brought by another haulage system (LHD, truck, rail) or a mobile crusher or feeder breaker which is located close to the stope and moved up at regular intervals. (Fig. 4-16).

Within these contexts, the major recent and ongoing developments in belt conveying are:

- Portable/mobile feeder breakers or crushers;
- Curved belt conveyors;
- High HP conveyors for use in steep inclines at high capacity, including a single drive for successive inclines;
- Booster conveyor drives; and
- Vertical conveyors.

(i) Portable/Mobile Feeder Breakers or Crushers

One of the major constraints to the application of this equipment in underground metals mines is the need for the equipment to be sufficiently low in height to permit movement throughout the production areas of the mine. This is now being overcome with the development of low height mobile crushers and feeder breakers. Horizontal jaw crushers are now available in semi-portable (2 m high) or fully mobile forms (height 2.7 m) at rates varying from

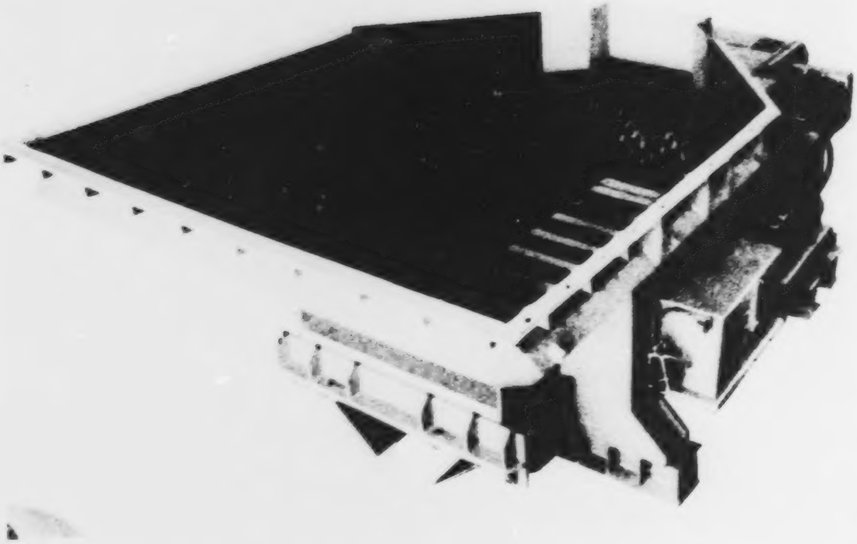
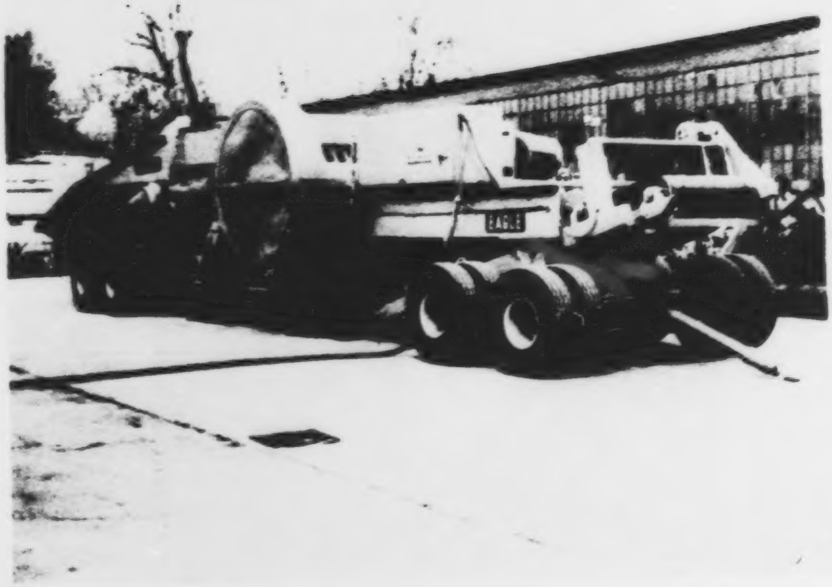


Fig. 4-16
Stationary primary
feeder breaker
(Courtesy Stamler)



Mobile primary crusher
(Courtesy Eagle)

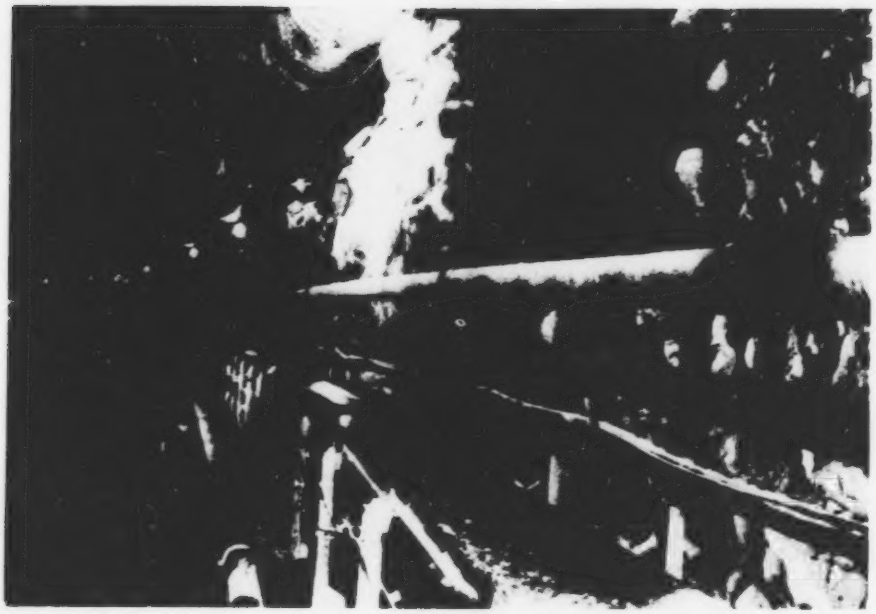


Fig. 4-17
Curved belt conveyors in
surface and underground situations
(Courtesy REI)

200 to 900 t/h, depending on rock hardness. Claims are made for crushing rock with strengths up to 2750 kN. The other form of mobile feeder breaker available utilizes a rotating drum or drums, carrying tungsten carbide tipped picks. These are working in granite, but with high pick costs.

A number of metal mines are investigating the use of portable crushers underground, in an attempt to take advantage of the benefits available with belt conveyors, namely:

- continuous high capacity haulage;
- reduction in manpower requirements, more so if automated or remotely controlled;
- increased safety;
- improved environmental conditions;
- reduced fuel bills;
- lower operating costs, if properly installed and maintained; and
- greater efficiency on inclines up to 25%.

Some form of crushing is essential for the successful use of belt conveyors underground, although the U.S. Bureau of Mines is experimenting with a run-of-mine rock conveyor.

(ii) Curved Belt Conveyors

Until recently, belt conveyors were limited to straight line operations with changes of direction in the horizontal plane being efefected by transfer points. Special types of belt conveyor (moulded pockets, etc.) are available for small output applications for vertical/horizontal movements, but any change of direction in the horizontal plane has until recently required a transfer station. However, developments in conveyor idlers and suspension techniques and in conveyor belting materials have now made it possible for belt conveyors to negotiate horizontal curves. There are now many installations of curved conveyors in the mining field, including underground mines. The minimum radius of curvature in the horizontal plane is usually about 200 m. Outputs up to 40,000 Tph are possible and lengths can exceed 10 km. (Fig. 4-17)

The availability of the curved belt conveyor considerably extends the range of applications of belt conveying in underground mining.

(iii) High HP Conveyors and Angle Stations on Steep Inclines

The improvements in belt technology now enable conveyor drives up to 10 000 kW and with speeds up to 30 km/h to be applied to underground mining. High outputs up steep gradients (1 in 4) are becoming commonplace in underground mines, either in declines from the surface, or from the lower levels of the mine to existing shaft bottoms.

An interesting development with the cable belt type of conveyor is the angle station. This permits the use of a single drive unit to power the conveyor which is travels up several inclines set at an angle to each other. The material is transferred from each sectional belt, but the wire cables carrying the belt are deflected by pulleys so as to continue up the next leg of the incline. This system is operating very successfully in a manganese mine in South Africa. (Fig. 4-18)

The main advantages are - elimination of drive units for each leg, no sequence control problems, no additional power cabling, and reduced personnel requirements.

(iv) Booster Drives

There is often a need to increase the output through an existing conveyor, or to lengthen an existing conveyor. One means of achieving this is to install in a predetermined position within the existing conveyor a booster or intermediate drive unit, which transmits power to the carrying belt by an endless driving belt which is in contact with the carrying belt. This imparts the additional power requirement to the original belt. These units, which can be installed at any point in the conveyor length, and in numbers to suit the increased capacity, have the following advantages:

- A new larger drive unit is not required;
- A new conveyor belt is not required since the belt tensions are kept lower;
- New delivery, return and tension units are not required as the tension is kept lower;
- As a result of the above, a considerable cost saving in capital expenditure is achieved;
- The booster drive can utilize the same components as the main drive; and
- Eliminates the need for additional transfer points.

(v) Vertical Conveyors

Special belting techniques utilizing conventional flat conveyor belting with additional special flexible rubber profiles and dividers now enable conveyors to be used to move material vertically at rates up to 1100 t/h. The flexible side wall permits the belt to bend in the vertical plane so that it can be loaded and discharged horizontally, and can be deflected to the elevating slope or shaft angle by deflection wheels.

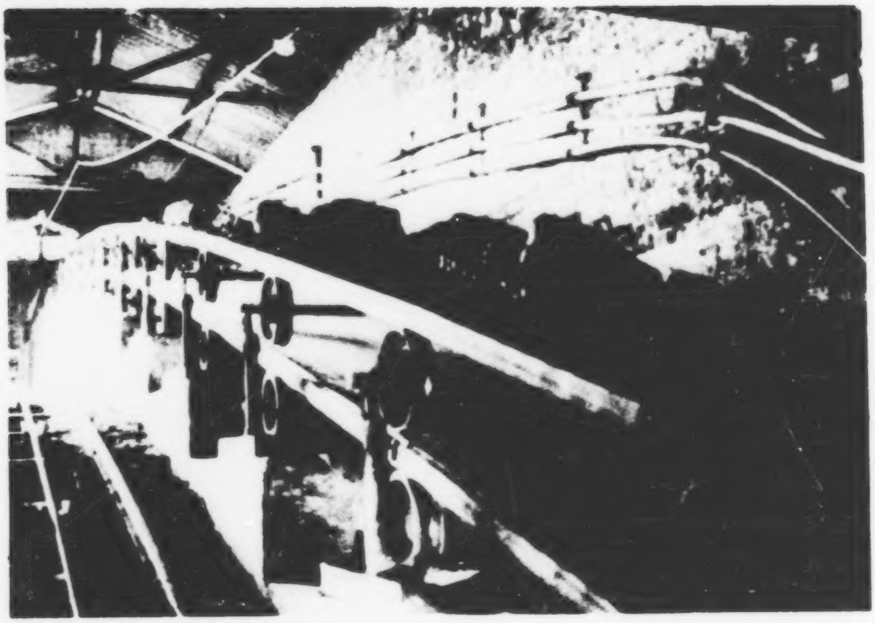


Fig. 4-18
Cable belt installation with single flight
in adjacent inclines (lower photo)
(Courtesy Cable Belt)

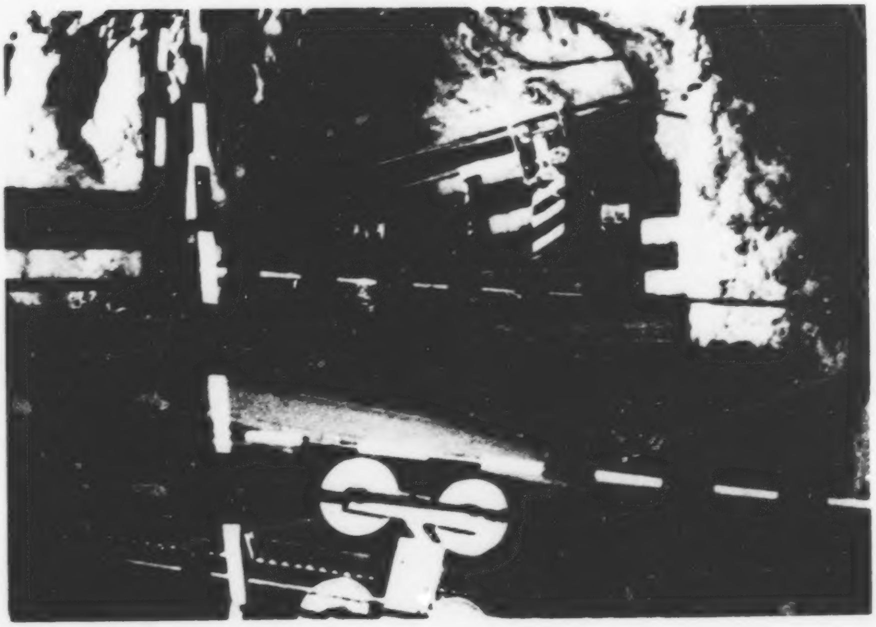
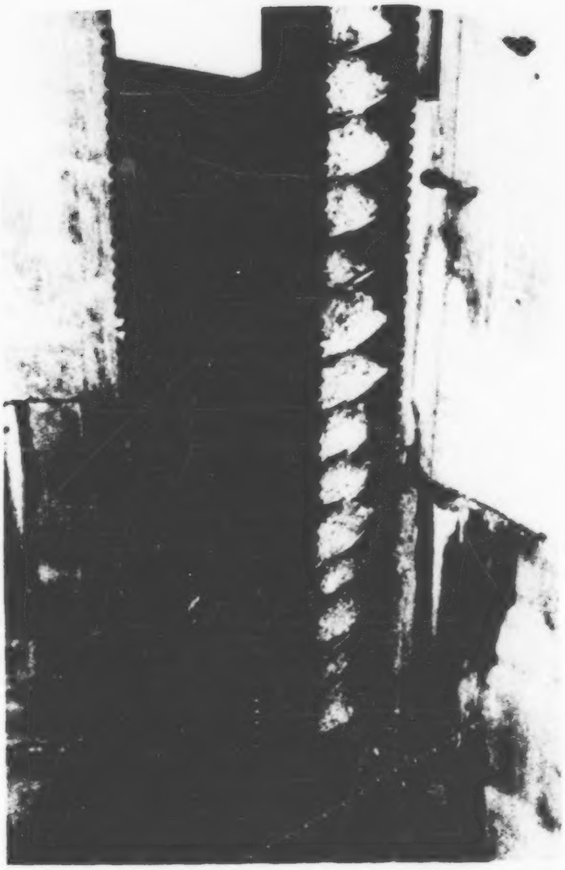


Fig. 4-19
Vertical conveyor in
construction shaft
(protective cover removed)
(Courtesy Flexowall)



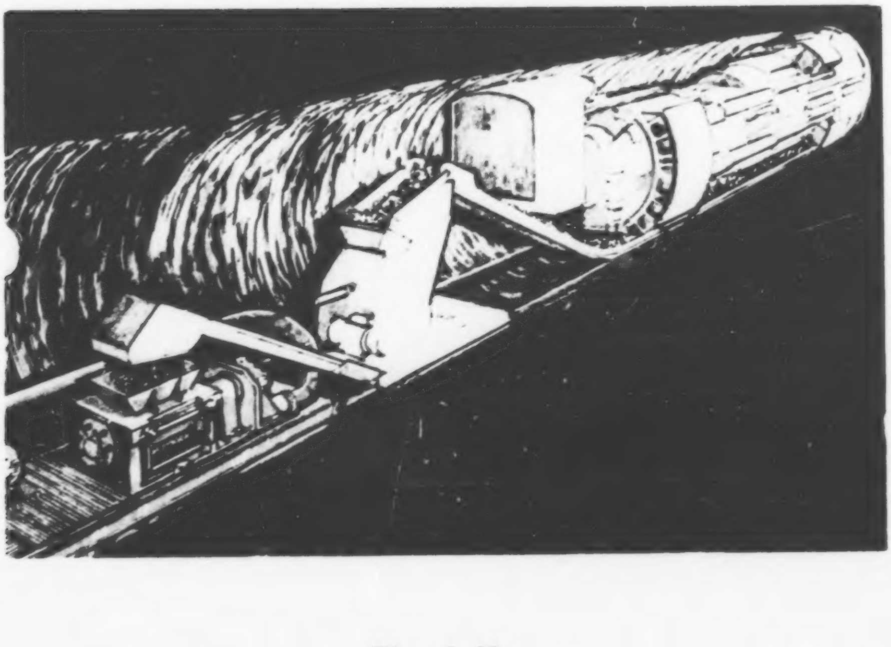


Fig. 4-20
Low pressure pneumatic
conveying system
(Courtesy Radmark)

Incline angles of up to 90°, capacities up to 3,500 m^3/h and vertical lifts of 370 m are now feasible. One conveyor is currently in use on a major tunnelling contract handling rock from a tunnelling machine at 700 t/h up an 80 m deep shaft. (Fig. 4-19)

The major advantages are:

- substantial savings in space;
- continuous haulage through shafts and steep inclines;
- lower maintenance and energy requirements;
- elimination of transfer points; and
- reduced capital cost.

.15 Pipeline Pipelines have been used for many years for solids handling in underground mines and almost exclusively for the transportation of back fill materials for stopes. These systems have been well documented and established and do not appear to be the subject of new developments.

However, in the past few years, the use of pipelines for the transport of minerals and rock vertically through shafts in both underground mining and construction work, is beginning to develop.

The most successful systems are those using low pressure air at high volumes. The system essentially comprises a low pressure blower (0.5 - 0.8 kg/cm^2), an airlock feeder and the pipeline. In mining, the system is being used to supplement existing shaft capacities, by handling minus 37 mm material through vertical lifts of 500 m at rates up to 200 t/h. This enables more ore to be hoisted by the existing skip system. Shaft capacities have been increased by up to 15% at a considerably lower cost than installing bigger hoists or skips.

The system is also being used successfully in tunnel construction, handling the output from tunnel boring machines, which is discharged directly to the surface through a borehole drilled specially for the purpose. (Fig. 4-20)

The advantages of the system are:

- low operating pressure;
- high air speed - no blockage problems in the event of a power shutdown - restart easy; and
- flexible system.

The potential applications for this system in underground mines seem to be:

(i) In some adit/drift mines with extended long haul workings could be alleviated by the raising of minerals through boreholes to the surface, thereby reducing or eliminating long underground hauls.

(ii) Older shaft mines sometimes have underground lower-horizon hoisting bottlenecks, which could be alleviated by raising the mineral from a deeper to an intermediate level by pipeline up a borehole.

(iii) Mineral conveyed by pipeline occupies less roadway and shaft space and leaves more effective cross-sectional area for men and materials.

(iv) Segregation underground may be necessary to avoid contamination and transport capacity restrictions in mineral clearance and in the surface treatment plant; therefore, it may be possible to separate development drivage and roadway repair debris, which can be transported independently by a pipeline system, either to the surface or to a backfill stope.

Development rock from new underground drivages at lower levels in the mine can either be separately raised to the surface or alternatively raised up a shaft or borehole to disused workings in upper horizons, combining the advantages of hoisting and backfilling.

.2 Transport of Materials and Supplies

Ideally materials transport should be accomplished using one type of vehicle between the stockyard/stores and the final destination, with no load transfers or haulage system changes.

Materials transportation underground in metal mines normally is undertaken by trackless or rail haulage. The former is almost entirely diesel powered, whilst the latter utilizes trolley and battery locomotives.

The major developments in this field are:

.21 Containerization It is proving more economical, and showing practical sense to load containers with materials and not rehandle the contents until they have arrived at their destination. This idea has recently been developed for use with the LHD type vehicle. The vehicle is fitted with a quick release device which enables the unit to attach to and to detach from a variety of units such as Bulk Materials Container, Manriding Container, Forklift, Self-Loading Container, and a series of speciality containers such as firefighting, pumping and workshop, etc.

.22 High Speed Locomotives As mine workings extend further from shafts, travelling time becomes critical. Where locomotive systems are already installed, benefit can now be made of new high speed diesel hydraulic locomotives - twin engines - 224 kW total, capable of speeds

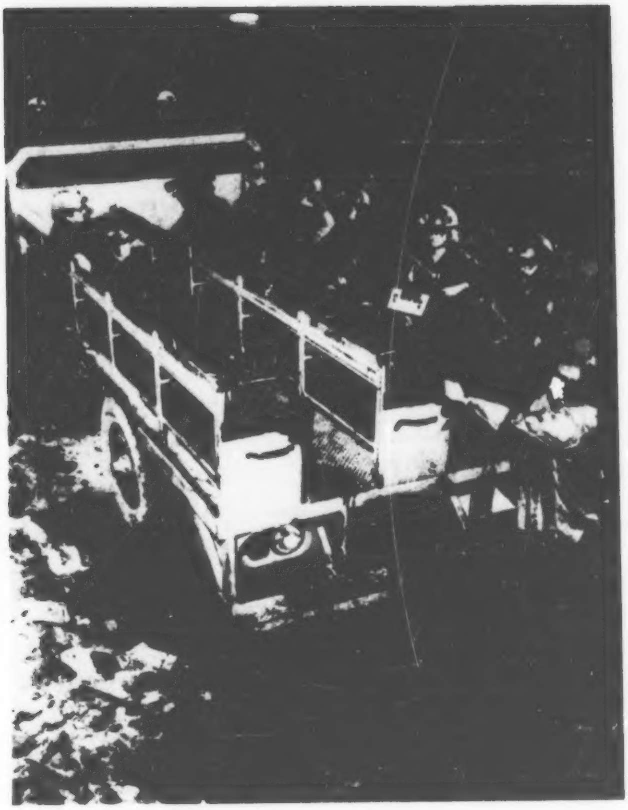
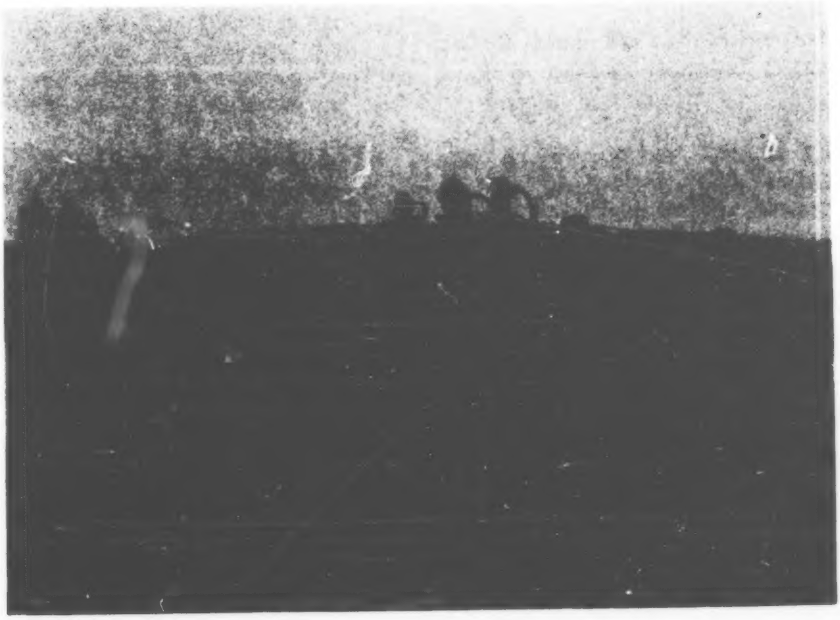


Fig. 4-21
Special purpose vehicles
- manriding; access platform;
lubrication
(Courtesy Teledyne)



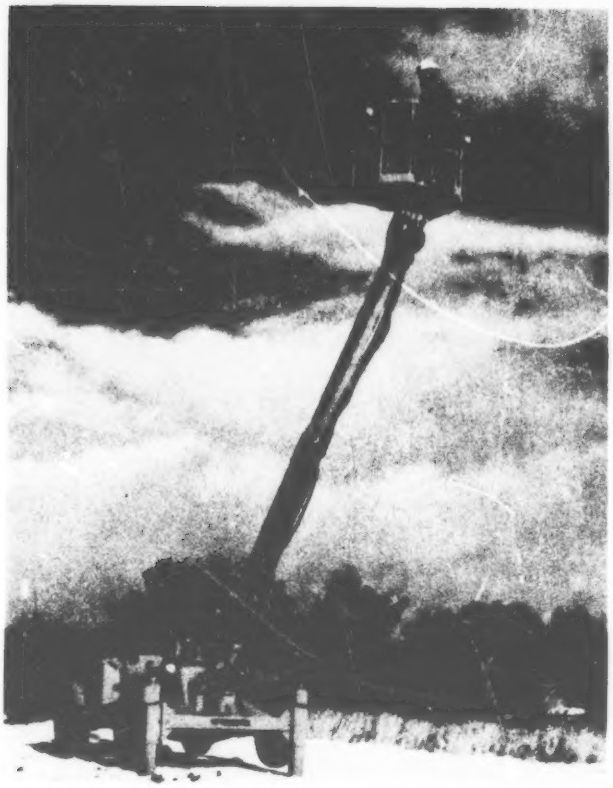


Fig. 4-21
Special purpose vehicles
- manriding; access platform;
lubrication
(Courtesy Teledyne)

up to 47 km/h equipped with fail safe brake gear. It goes without saying that very good standards of track installation and maintenance are a necessity for travel at these speeds.

.23 Monorail Haulage The use of monorail systems is beginning to find favour in some European mines, operating either with diesel locomotives, or utilizing rope haulage. The biggest advantages they offer are:

- electric powered rope haulages;
- less track maintenance;
- can be used with trackless mining systems;
- can handle heavy, awkward loads better than rail or trackless methods; and
- can be used on severely undulating roadways.

Since many metal mine loadways meet the last condition, it is difficult to understand why there is not more use made of these systems for transporting men and materials in Canadian mines.

.24 Rope Haulages The main developments in this field have taken place in European coal mines, but they may have some application to underground metal mines with undulating or directionally varying roadways.

Traditional types of rope haulage engines are mainly on line electrical motor driven with clutch or variable fluid coupling and rotor resistance types. Squirrel cage motor control systems have now been developed with "soft start" and "cycloconverter" using thyristors.

A form of trapped rail system is used to captivate both the vehicles and the rope through curves of five metres minimum radius and through changes in gradient.

The vehicles utilize a bogie system which incorporates a novel system of braking suspension and rope guidance. The bogies are designed to eliminate the radial rope pull effect on curves by having a side wheel which runs in track side channels on curves and at undulations. These channels also provide full captivation of the bogie and assist in turning the bogie through the curve.

This controlled rope haulage system could have applications particularly where entry into the mine is by inclined drift equipped with rope haulage.

.25 General There is a noticeable trend in the field of trackless vehicles to produce more special purpose vehicles than general purpose types. Vehicles are now available for such duties as Anfoloaders, shotcrete units, scalers, mobile breakers, scissor lifts and service and lubrication vehicles, to all of which the low profile concept is now being applied. (Fig. 4-21)

.3 Transport of Men

The two main methods of transporting men to and from their place of work are trackless vehicles and rail systems. The developments described in the two previous sections are also mostly applicable to man-riding systems.

Locomotives are transporting 240 men at one time up inclines of 6%. The locomotive and carriages are fitted with spring applied brakes held off by compressed air generated by the locomotive. (Fig. 4-22)

These brakes and those on the locomotive are so arranged that they may be applied at any time by either the driver or the guard. Brake application is automatic when any part of the train breaks away from the locomotive.

Development of the "leaky" feeder radio communications system gives the facility of signal/speech from a moving train or haulage.

Apart from improvements in these vehicles, there appear to have been no major developments in this field in metal mines.

Two methods used in Europe are worth mentioning briefly: belt conveyors and, in steep drifts, chair lift (ski-type) systems (Fig. 4-23). Each system provides continuously available transportation in two directions.

Fig. 4-22
Transport of men by
locomotive on steep inclines
(Courtesy Hunslet)

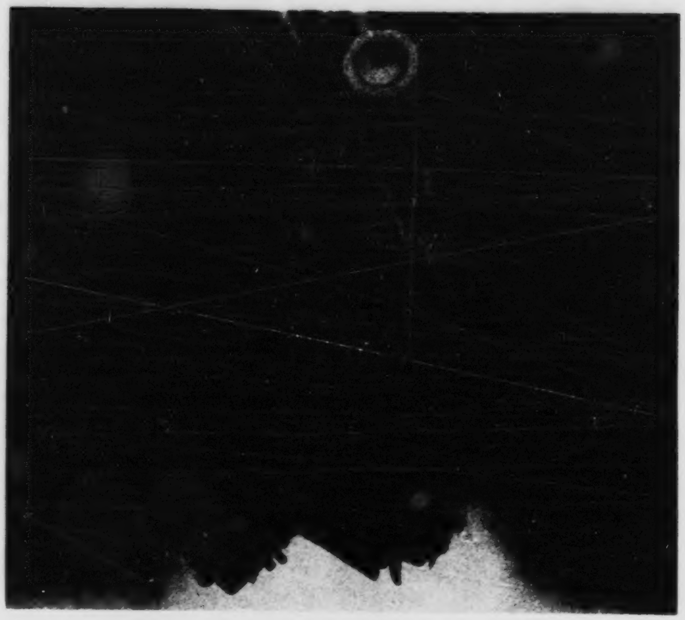


Fig. 4-23
Transport of men by ski lift
(Courtesy Schwarz)

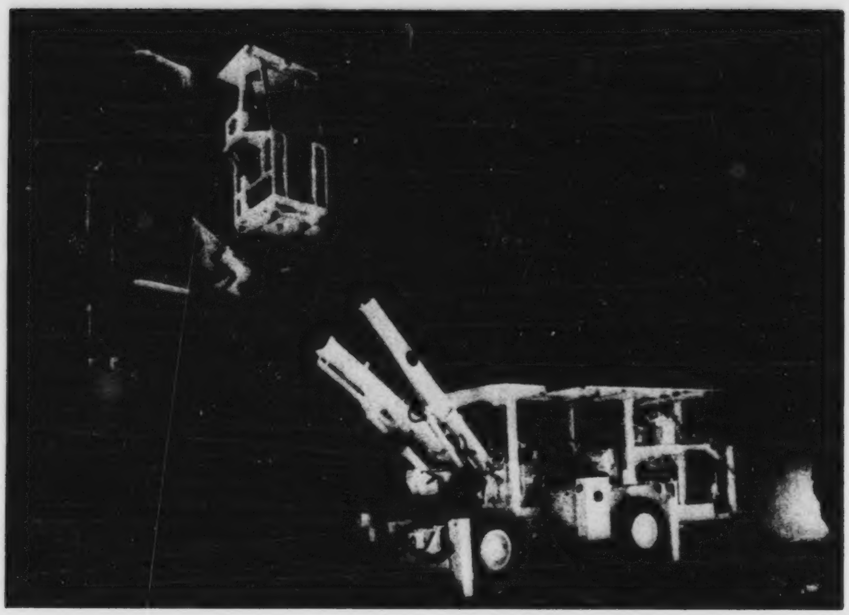


Fig. 4-24
Electric powered remote controlled rock bolter
(Courtesy SIG)

Fig. 4-25
Close up of rock bolter showing carousel
(Courtesy SIG)

4.5 SUPPORT SYSTEMS

451 BACKGROUND

In most forms of underground mining, some system of support, permanent or temporary, is required to allow safe extraction of the ore. Ground support is the controlling factor in all phases of mine design, and the limits to the size of excavation, stoping method or breaking of ground, become more critical as the depth of mining increases.

Traditionally, timber has been a major source of support material in underground mines, especially in Canada where it is a low cost and readily available product. The alternative to timber became the steel girder or arch where permanent and/or greater load bearing capacity support was required.

The 1950's saw the beginnings of the use of rock bolts as systematic support, as compared with its earlier uses for random pinning of loose rock. The 1960's saw the fully grouted bolt introduced. The rock bolt, in its many and varied forms, used with or without mesh is probably still the most common support system in use today in underground metal mines. Estimates indicate that over 500 million are used annually worldwide; and over 50 million per year in Canada. Types available have developed from the mechanical through the grouted (cement or resin) to the full length friction bolt and other types. These are referred to in more detail below.

Other methods used to support exposed rock surfaces and permit passage for services, etc., are:

- sprayed concrete (gunite, shotcrete, etc.) with or without the use of mesh;
- mass or reinforced concrete;
- injection of cement or chemical materials into the ground for stabilization (grouting); and
- yielding supports such as hydraulic chocks.

In addition to the above, the other methods in common use to provide support in stopes and other mining areas are:

- back fill material - waste rock, sand, cemented tailings, etc.
- naturally caved waste

The developments and trends in support systems for underground are generally taking place in two areas - the support devices and the systems used to install them. These are examined in the following section.

452 DEVELOPMENTS AND TRENDS

The major areas where developments are taking place are:

- fully grouted rock bolts - grouting materials;
- full contact friction rock bolts;
- fibreglass bolts;
- cable bolts;
- shotcrete;
- fully mechanized bolting equipment; and
- mechanized shotcreting equipment.

.1 Fully Grouted Rock Bolts

Over the past four years, the rise in the use of resin bolts has been quite significant. Resins with various rates of hardening are now available, thus enabling bolt installation times to be reduced. However, since the resins are partially oil based products, the cost reflects the oil price with a consequent increase in the cost of resin bolting, and at current trends, it is unlikely to be economically competitive with ordinary mechanical rock bolts. Resins are also flammable and represent a potential hazard underground. The advantage of full length support, however, offsets this apparent economic disadvantage.

One competitor in this field, the cement-grouted bolt, whilst considerably cheaper, suffered from lack of control over the grout consistency (through variable added water quantities) and hence in the strength of the bolt/cement/rock adhesion.

However, recent developments have altered this state of affairs. A new cement anchor has been developed which is supplied in cartridge form. The cartridge contains cement with the water in a microencapsulated form. The microcapsules vary in size from 1000 to 1700 microns and are mixed with the dry cement when the insertion of the bolt ruptures the capsules. The mix is even and produces a strong quick set of the bolt in the roof. Because the water is released uniformly across the hole, the homogeneous grout then formed hardens rapidly to provide firm full length anchoring. Full load setting times are a few minutes.

By varying the cement type and additives, grout strengths and setting and load bearing times can be varied.

The system is completely safe, non-toxic, non-flammable and non-allergenic.

It is the first anchoring system to have the precise amount of water in a microencapsulated form already mixed with quick setting high strength cement. It is expected to cost at least 20% less than the resin type cartridge.

There is no doubt that this new system offers distinct advantages and should find widespread application in underground metal mines.

.2 Full Contact Friction Rock Bolts

Mechanical rock bolts, whilst cheap, have the major disadvantage that the anchorage is at a single point in possibly unknown conditions. Resin and cement grouted bolts offer full length anchorage and strata bonding but are more expensive.

Recently, two developments in rock bolts have attempted to combine the full length bond of the grouted bolt with the ease of setting and lower cost of the mechanical bolt.

There are currently two types of these new 'friction' bolts available - the split set bolt and the swellex bolt.

The split set bolt is a pipe split along its length, nominally 38 mm diameter, which is forced into a 35 mm drilled hole by impact or percussive means. The borehole serves as a die and the split set bolt is force fitted into the hole. Bolt insertion forces are normally between 2.5 and 5 tonnes per metre/length of the bolt. After setting, the force necessary to dislodge the bolt is almost the same as that required to set it, but the anchor force increases with time due to lockup from ground movement and corrosion. The split set bolt is substantially less costly than grouted bolts and only slightly more costly than mechanical bolts. The same equipment is required to install the bolt, and no additional action is required thereafter. The split set permits substantial movement in the ground while still maintaining roof support.

The split set bolt has been available for over four years and the annual usage has reached approximately 4 million.

The swellex bolt consists of a 41 mm diameter steel tube, the outer diameter of which has been reshaped to assume a smaller (28 mm) external diameter. The ends of the tube are reinforced with short sleeves sealed by welding, one of the sleeves retaining the plate washer. A small hole is drilled through the sleeve to allow water to be injected into the bolt during installation.

The bolt is installed by placing it in the 38 mm diameter drill hole - hand pressure is enough - and high pressure water (300 bar) from a small pneumatic or electric pump is pumped into the bolt, causing the bolt to expand to follow the irregularities of the drill hole thus increasing the friction forces. During this swelling process, which takes about 20 seconds, the length of the bolt is reduced through contraction, resulting in the plate washer being firmly tensioned against the rock.

Also, since the hole diameter (38 mm) is less than the original pipe diameter (42 mm) a tongue is left inside the expanded profile. When the water pressure is released, the tongue acts as a spring, producing radial forces on the bolt which continues to press against the walls of the hole.

The system has a number of advantages over the split set bolt:

- Does not require machinery to install the bolt in the hole, therefore, larger bolts can be installed in a given height of roadway;
- Better friction wall grip is obtained since the tube contours the wall; and
- Hole diameter is less critical, 28 mm bolts can be set in the range 30 mm to 39 mm without loss of friction grip.

In addition, bolts can be rapidly installed close to the face to provide immediate support. The method lends itself to installation by automatic rock bolting systems.

The swellex type of friction bolt would appear to offer more consistent loadings in conditions where hole gauge is a problem, and a simpler means of installation.

.3 Fibreglass Bolts

There are several applications for rock bolts in which the use of steel for the bolt can cause problems. This is especially true where the support to the ground is temporary, i.e., until the rock/ore is blasted down as in the cut and fill stoping system where the roof to each successive lift is bolted. It is obviously not desirable for steel bolts to get into the crushing system.

Success has recently been achieved by substituting fibreglass for the steel in the bolts. This still provides adequate support to the roof and removes the 'steel in crusher' problem. The fibreglass can also be drilled through without damage to the drill bit.

.4 Cable Bolts

The need to presupport ground before it is mined by certain mining methods such as shrinkage or cut and fill stoping (to prevent dilution or to provide immediate support to the newly exposed rock after blasting) has led to the development of the cable bolt and truss form of support.

Holes up to 20 m long are drilled into the back of the stopes, or into the rock outside the intended excavation limits. High strength cables are placed in the borehole together with a breather tube and a cement grout is pumped in under pressure, thus grouting in the cable. Facilities can be incorporated to tension the bolt once the cement grout has reached its full strength.

Cable bolts were first used in the early 60's, mainly in the civil engineering industry, and it is only in the past few years that the technique has begun to be more widely applied in the Canadian mining industry, particularly in the field of extended ground support. Its use has enabled more open forms of mining to be practised such as cut and fill in place of square set stoping, it has provided better roof support in working stopes thus improving safety, and it has provided support to the stope walls to reduce or prevent spalling or dilution.

One problem with the cable bolt is the delay period involved before the cement grout sets and the cable becomes load bearing. Attempts have been made to develop a mechanical anchorage to provide an early load, but these are not believed to have been too successful, particularly from an economic standpoint.

The present trend, however, seems to be to use the cable bolts on a wide pattern interspersed with some type of rock bolt. The rock bolt thus provides the immediate support whilst the cable bolts provide the extended ground support.

.5 Shotcrete

Also known as gunite or sprayed concrete, this technique has been in use for many years in both the mining and construction industries. It consists essentially of spraying a concrete mixture onto the exposed rock face where steel mesh may or may not have been placed. The earliest methods used a dry mix of aggregate, sand and cement with water and any additives (e.g., quick setting) being added just prior to discharge from the nozzle.

More recently, wet mix shotcreting has attracted more attention due to the better quality control available since the water is added through meters at the mixing chamber.

The latest development in this field is the production of a very high strength concrete by adding steel fibres to the mixture. The resulting concrete is called Fibercon, and it has improved resistance to fragmentation, spalling and to cracking. Improvements of up to 600% over non-reinforced concrete are claimed in impact strength and spall resistance.

With the advent of automatic placing equipment (dealt with below), shotcrete is finding more application in mining. Cost savings of up to 20% have been achieved, compared to timber supports.

.6 Rock Bolting Equipment

The level of mechanization of rock bolting is low compared with other kinds of work underground, despite the fact that every year, huge quantities of bolts are installed. The rock bolting procedure should be easy to mechanize and automate.

Obvious efforts are now under way to mechanize and automate this process. The requirement is for equipment that can drill and place rock bolts as soon as possible after excavation, and without the operator having to enter the unsupported area.

Early developments utilized the mechanized drilling jumbo to perform the drilling operation, but the heavy duty work of inserting the bolts (and resin or grout) remained. With labour being a high proportion of bolting costs, it is difficult to understand why full mechanization has taken so long.

The Europeans were the first to develop mechanized rock bolting rigs in the 1960's but it is only in the last three or four years that serious work has been done to produce the fully mechanized rock bolting unit. (Fig. 4-24)

These units usually comprise four major elements carried on a turret mounted on the end of a universal boom. These four elements comprise the drilling unit (usually hydraulic), a charging unit (for inserting resin or cement capsules or grout), a carousel holding from 4 to 12 bolts, and a bolt insertion and setting unit. (Fig. 4-25)

The most common operating system is that in which the operator controls each phase remotely from the control panel, selecting each phase in turn. In this way, several bolts can be installed without the operator leaving the control point. Machines can be adapted to install mechanical, grouted or friction type bolts. Injection of the grout can be by capsule fed into the charging system manually by the operator, or by the most recent development whereby the resin and catalyst, or the cement and water are held in separate tanks and pumped through two concentric pipes into the hole where they mix.

The most recent development permits fully automatic operation of the bolting cycle. The operator positions the boom, and initiates the cycle. The machine automatically drills the hole, injects the grout if required, inserts the bolt and sets it. Automatic operation then stops. Whilst the operator is manoeuvring the bolter to the next hole, a new bolt is inserted into the bolt rotator from the carousel ready for the next cycle.

Fig. 4-26
Mobile shotcrete placing machine
(Courtesy Eimco)

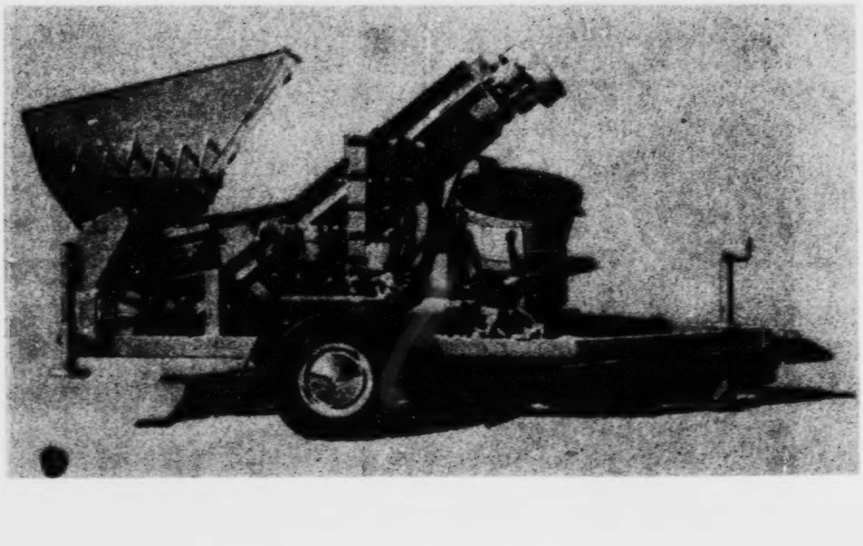


Fig. 4-27
Mobile cyclone type
dry mix shotcrete unit
(Courtesy Eimco)

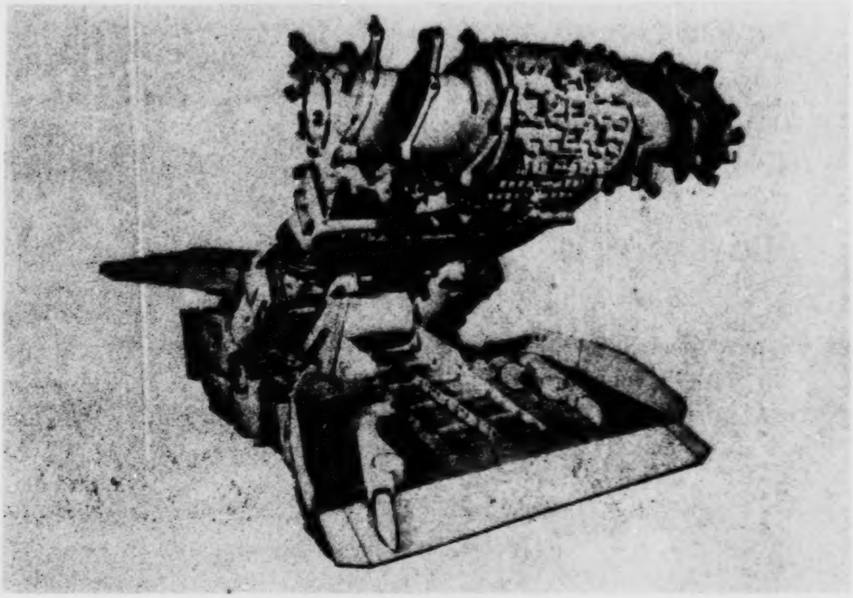


Fig. 4-28
Typical continuous miner
(Courtesy Joy)

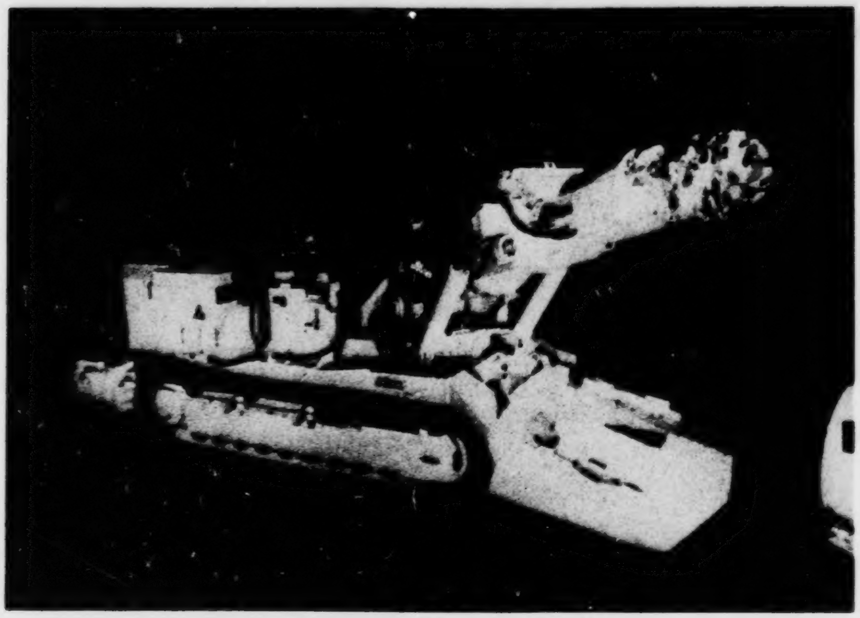


Fig. 4-29
Typical roadheader
with 112 kw cutting head
(Courtesy Anderson Mavor)

It has been found that mechanized bolting can reduce bolting costs by as much as 40%. Figures for the fully mechanized system have yet to be obtained, and because of the higher capital cost of the equipment, savings will depend even more on the degree of utilization of the machine.

There is no doubt that considerable improvements in cost, productivity and safety can be achieved with mechanized rock bolting units, and that there is tremendous potential for this equipment in metal mines. Coal mines in the U.S.A. have been demonstrating the advantages for many years.

.7 Shotcrete Placing Equipment

The traditional method of placing shotcrete has been to use a small mobile shotcrete unit feeding through a hose to a nozzle held by an operator, a second man loading the mix ingredients into the machine hopper. This is still a favored method where small quantities are to be placed, but in situations where large or regular applications are necessary, the mining industry is beginning to utilize the mobile, mechanized equipment which has been in common use in the construction industry for many years.

The equipment now available to mining includes:

- Remote controlled booms which allow the operator to place the shotcrete from a remote and safe control point (Fig. 4-26);
- Mobile vehicles (rail or trackless) which incorporate receiving hoppers, wet or dry mixing units, charging feeders and additive feeders (Fig. 4-27); and
- Shotcrete transport vehicles (rail or trackless).

Systems for mining are designed with capacities of up to 8 m^3/h of concrete with a compressive strength of 340 bar.

The availability of the remotely controlled boom will bring the greatest benefit to the mining industry since it removes the operator from the nozzle area to a safer and cleaner working environment.

One aspect of the rock bolting system has yet to be successfully mechanized. This is the handling of the steel wire mesh, or of flat steel bars which are often used to provide additional areal support to the rock bolt.

To date, no serious attempts appear to have been made in this field, one which is necessary to be able to fully justify the use of automated rock bolting, and to remove the operator from working in an unsupported area.

4.6 ACCESS EQUIPMENT (MECHANIZED EXCAVATION)

461 GENERAL

The aspects which will be considered in this section are the developments and trends in mining equipment for the mechanical excavation of shafts, raises and tunnels and slopes in mines.

462 TUNNEL DRIVAGE

There are usually three major reasons for considering the use of machines to completely excavate roadways or to produce ore in mines. These are the lack of skilled miners, higher performance capabilities and better ground control.

There are three general types of machine available for this task - continuous miners, roadheaders and full face tunnel boring machines (TBM's).

.1 Continuous Miners

These machines were developed for and are in wide use in coal mines throughout the world. Their non-coal application is, however, limited because of their dependence on high speed tungsten carbide tipped cutting picks to the softer minerals, such as low grade iron, gypsum, potash, trona and salt, with compressive strengths up to 350 kN. They are not, therefore, applicable for metal mining. (Fig. 4-28)

.2 Roadheader

The roadheader can be loosely described as a loading machine equipped with a cutting head mounted on the end of a boom, and applied to the rock face in such a manner that only a small portion of the face is cut at any one time. These machines also utilize tungsten carbide tipped picks but at considerably lower speeds than continuous miners.

Earlier machines with up to 90 kW available to the cutting head were found capable of cutting low abrasive rocks and ores with compressive strengths up to about 600 kN, but the most recent generations with cutting powers up to 220 kW can cut rocks with compressive strengths up to 1000 kN but again, in the interests of cutter costs, only rocks with low abrasiveness. (Figs. 4-29 & 4-30)

Recent developments concern the addition of high pressure (700 bar) waterjets to assist the cutter picks. This has dramatically improved the performance of the machines and could lead to an extension in their application to hard rock mining.

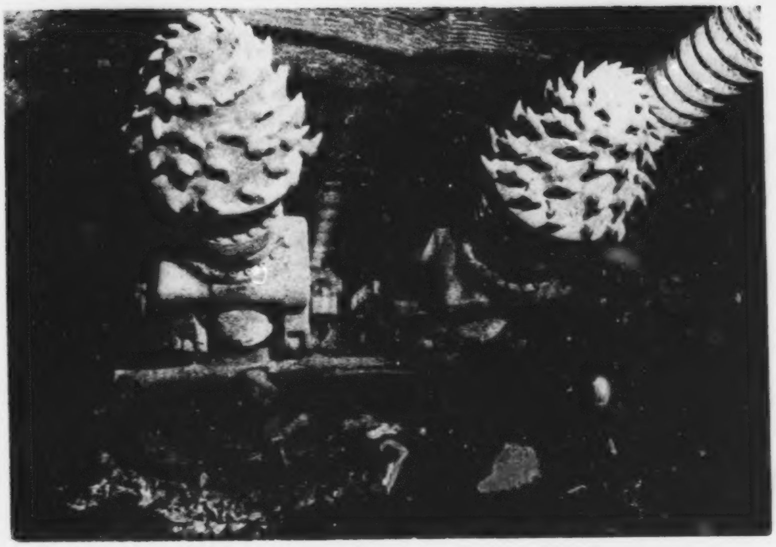


Fig. 4-30
Twin boom type roadheader
(Courtesy Dosco)

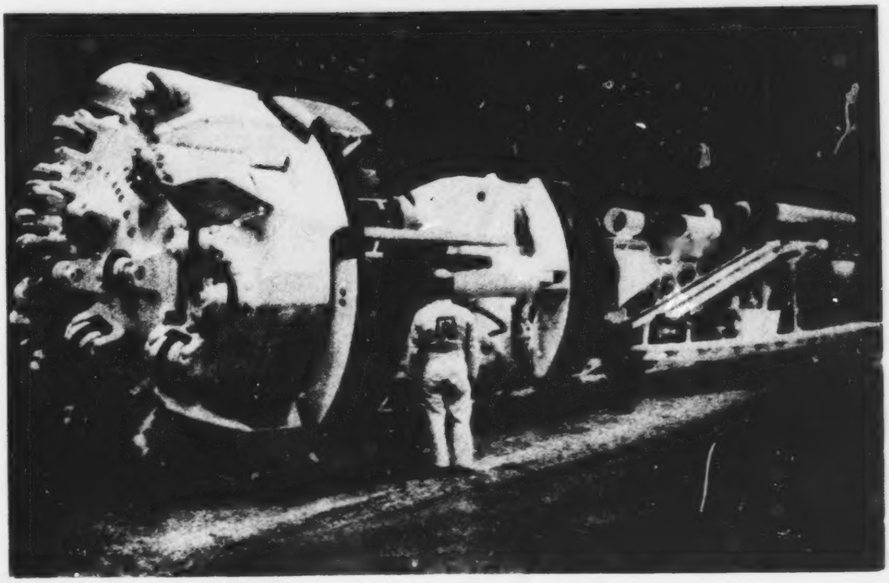
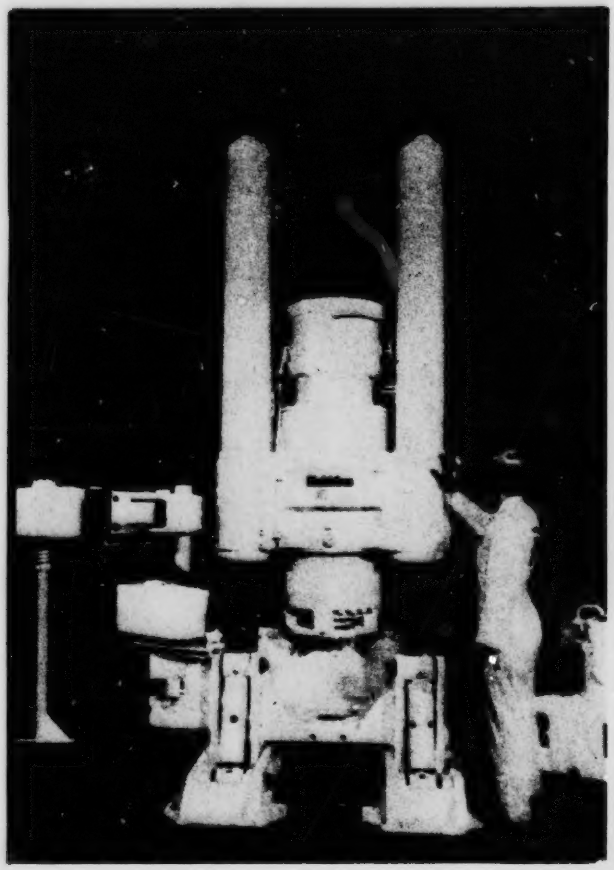


Fig. 4-31
Tunnel boring machine for 3.35 m dia.
drivage in gold mine
(Courtesy Robbins)

Fig. 4-32
Typical raise boring machine
(Courtesy Robbins)



Several of the first generation machines have been tried in metal mines (lead, zinc, copper, silver) in the past five years, but were unsuccessful due to high cutter costs and low productivity in relation to capital cost.

This type of machine offers the advantages of good ground control emanating from a cut roadway profile, good access to the face, and relatively good mobility and flexibility. However, its inability to economically excavate hard abrasive rocks will probably limit its application in metal mining to the 'softer' ores such as the borate ores colemanite and ulexite. In the Southwestern USA second generation machines are being used to produce these ores from cut and fill slot stopes, and to mine oil shales.

.3 Tunnel Boring Machines (TBM)

The TBM is a machine that excavates the full face of the tunnel. It almost always cuts a circular section, utilizing a rotating cutter head carrying picks, discs or button roller cutters according to the hardness and abrasiveness of the strata to be cut. These machines, which have been used in the tunnel construction industry for over 25 years, can now successfully and economically excavate strata from the softest unsupporting clays and silts to the harder quartzites (3600 kN).

The first tunnels to be driven by TBM's were in 1952 for a construction project. The first TBM tunnels in a metal mine were driven in 1957. Since then, the four major manufacturers of TBM's have driven over 450 tunnels (at Jan. 81) of which only 42 (or 9%) have been for mining companies. Of these 42, only 18 have been for metal mines.

Mining applications have included inclined access drifts both from surface and between levels, main development drivages, ventilation tunnels and main haulage tunnels (rail and conveyor). Sizes have ranged from 2.13 m to 6.5 m diameter, gradients from +17° to -27°, lengths from 137 m to 19 000 m and rock hardnesses from 140 to 3030 kN. (Fig. 4-31)

The TBM is a specialized tool which has the capability and flexibility to offer, in the right circumstances, high rates of excavation, ability to excavate a variety of materials, a stable opening of circular section plus the ability to set support close to the cutting head, labour saving potential and a safer working environment. It is for these reasons that TBM's have gained such wide acceptance in the civil construction industry.

There are, however, several disadvantages to be borne in mind. The capital cost of a TBM is high, such that a minimum distance for drivage is essential to reduce the fixed costs proportions to acceptable levels. The time taken to erect the machine underground, or to dismantle and re-erect in another section of the mine can be quite considerable. There must also be sufficient potential drivage available to enable the machine to be fully utilized. As a very rough guide, tunnels less than 2 km in length do not individually justify the use of a TBM.

An excellent example of the use of a TBM for mine development can be found in a German coal mine. The seams are steeply dipping and long drivages in sandstone and schist are necessary for haulage, access and ventilation roadways. Since 1974, a TBM has driven six mine roadways at a diameter of 6 m, the individual lengths being 2.64 km, 2.89 km, 2.72 km, 2.09 km, 2.75 km and 8.0 km*. At the end of each leg, the machine was dismantled, transported and rebuilt for the next drivage. Each reposition took an average of four months. The average advance rate, including repositioning has been in excess of 2.2 km per year.

The modern tunnelling machine offers a great potential to the metal mining industry, and can, in the next decade, be expected to make an impact in areas where long drifts, inclines, declines, or cross-cuts are required quickly, and especially help to produce the fastest return on investment by enabling the mine to reach production earlier than by using conventional means of drivage, since TBM's can develop roadways between 3 and 10 times faster than conventional means.

It is also worth recording that gold mines in South Africa are beginning to introduce TBM's as part of a mechanization program aimed at improving productivity and increasing rates of advance. The machines are being used for strategic tunnels such as main intershaft connections and exploration and haulage tunnels.

The most interesting development in TBM's over the last few years, apart from the continuing development of the machine technology is the application of the machines for the drivage both upwards and downwards of steeply inclined roadways. The maximum angle achieved to date is 44°. This considerably extends the possible range of application of the TBM to mining operations.

* under construction

463 RAISE BORING

Raise borers are not new to the mining industry, and are becoming more accepted as developments have brought about advances in capacity (up to 6.4 m dia., and 600 m length), lower machine profiles, better reliability, variable speed drives to meet different rock conditions, and a trend to lower operating costs.

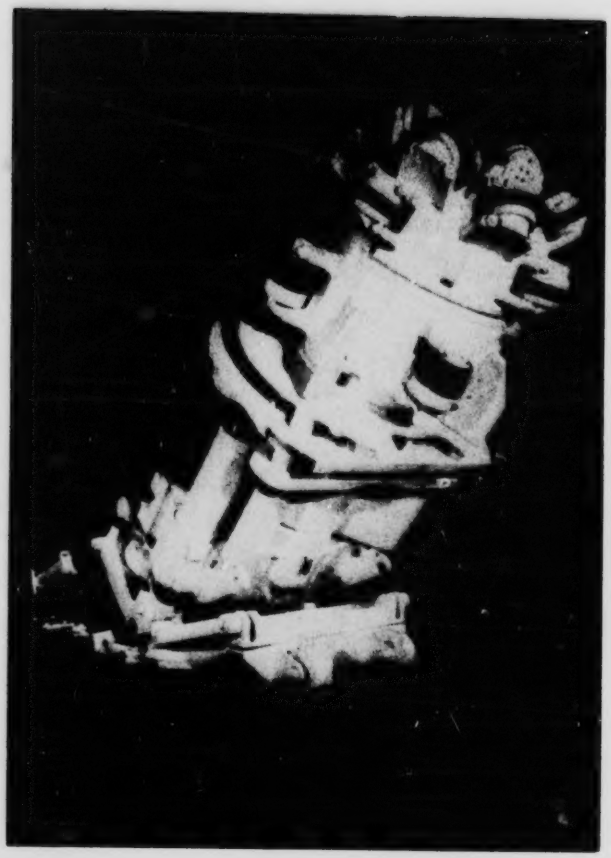


Fig. 4-33
Blind raise boring machine
(Courtesy Robbins)

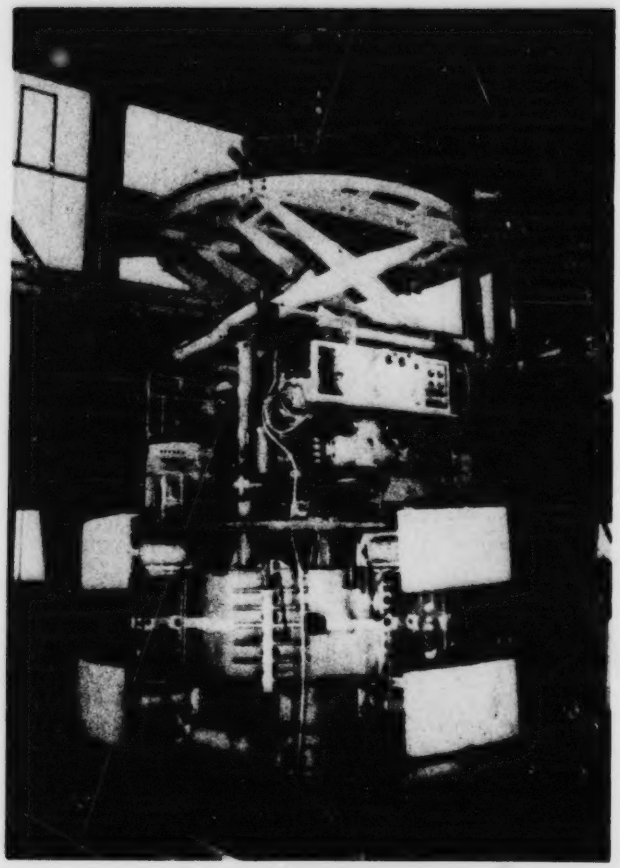
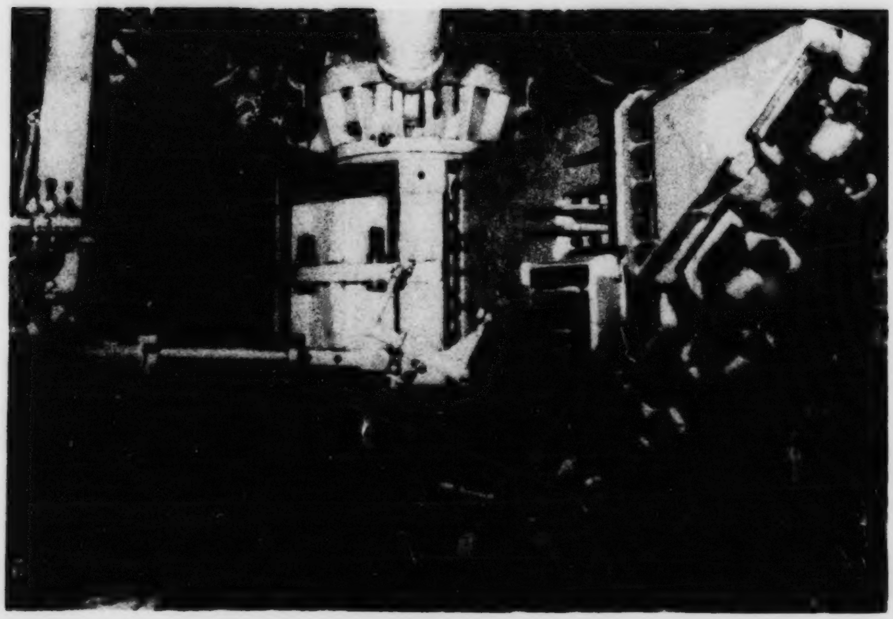


Fig. 4-34
Shaft borer with section
of cutter head
shown in lower photograph
(Courtesy Wirth)



The raise borer is being applied to a much wider field, where in addition to the normal ore, waste and ventilation raises, it is now forming service raises, stope access raises, slot raises, fill raises and even pumping raises where the raise acts as the pipe. Raise boring machines are also being used more frequently to drill shafts from the surface to underground workings, or to provide pilot holes from shaft bottoms to lower levels of the mine to assist with shaft deepening operations. (Fig. 4-32)

The pilot hole drilling capacities are also being used to drill small diameter drainage, sandfill or even power cable holes.

Recent trends in standard raise boring machines are to the use of single pass cutting heads rather than multi stage reaming heads (which suffered from higher torque requirements, gauge loss and rough running), and the use of computers and microprocessors to control bit weight, rotational speed and penetration rate to achieve optimum performance allied with acceptable cutter costs.

The most recent development, however, has been the introduction of the blind raise boring machine (or boxhole borer). (Fig. 4-33) This technique does not require a pilot hole, nor access to the termination point of the hole, the hole being drilled in one pass. The normal method is to bore from the lower level to the upper level, the cuttings being collected in a hopper above the machine. However, blind hole raises can now be drilled from the top using a double annulus drillpipe and reverse circulation air or water for muck removal.

There are two distinct types of machine available, one utilizing a rotating drill stem with the driving and thrust systems located at the starting point of the hole, the other using a non-rotating drill stem, the drive motor and gearbox being located immediately behind the cutterhead. In both cases, the stabilization of the drill stem is of paramount importance.

Blind bore raise machines have so far not been used on raises longer than 100 m, nor over 2.2 m diameter. The method is still in its infancy but can be expected to have increasing application for the mining industry.

By early 1981, there were over 300 raise drilling machines located in 25 countries throughout the world. Only 15% of these machines were in use in Canada, a low figure considering Canada's standing as a metal mining country. This is in line with the survey results, which indicate that international mines make much more use of the raise borer than do Canadian mines. The opportunities in Canada, therefore, should be good.

464 SHAFT BORING

This is a technique has been developed in the past 10 years, but more especially in the past 3 to 4 years.

Shaft sinking using the conventional basic 'drill, blast, muck' cyclic approach is probably one of the most time consuming, costly and hazardous operations associated with the development or expansion of a mine. Whilst there have been improvements with drill jumbos and mechanized mucking, it is still a labour intensive operation in a poor working environment. This system of shaft sinking rarely exceeds 15 m per week for diameters over 6 m.

However, equipment has been developed to enable mine shafts to be mechanically bored. This equipment can be of one of four forms.

.1 Raise Borers

An extension of the raise drill principle whereby shafts up to 6 m in diameter have been formed by drilling a pilot hole and reaming to full diameter in a single or multiple stages. This method has its limitations since access is required to both top and bottom of the hole.

Using this equipment, shaft sinking rates of up to three times conventional rates are being achieved with considerable savings in cost and time, and increased safety.

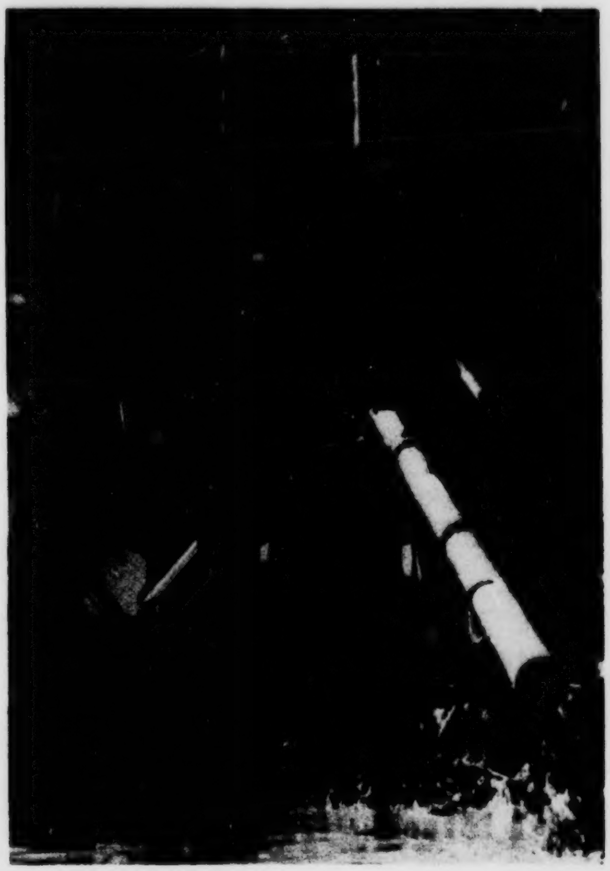
.2 Shaft Borers using Pilot Holes

This method, developed and now used extensively in Germany, requires a pre-drilled pilot hole, with the main shaft excavation being carried out by a vertical boring machine (VBM) located in the shaft, and independent of the surface for drive. The unit essentially is a Tunnel Boring machine turned through 90°. Cuttings from the cutting-head are scrapped towards the centre of the shaft and fall to the bottom of the pilot hole. Access is thus required to both top and bottom of the hole. Within certain limits, deviation of the pilot hole does not affect the verticality of the shaft (as could be the case with the raise drilled shaft) since the VBM is steerable. (Fig. 4-34)

To date some 20 shafts have been sunk by this method in Germany. Diameters range from 4.5 to 7.2 m, depths from 106 to 463 m and average sinking rates from 4.1 to 13.5 m/d (max. rates 8.5 to 30.1 m/D).

In Alabama, a 7.0 m diameter 600 m deep shaft has recently been started using this method.

Fig. 4-35
Blind shaft drilling unit
(Courtesy Hughes)



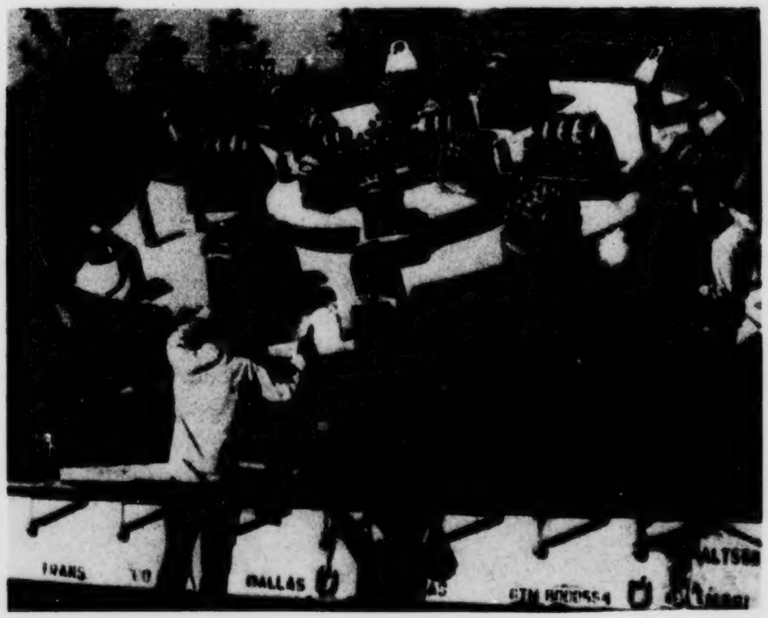
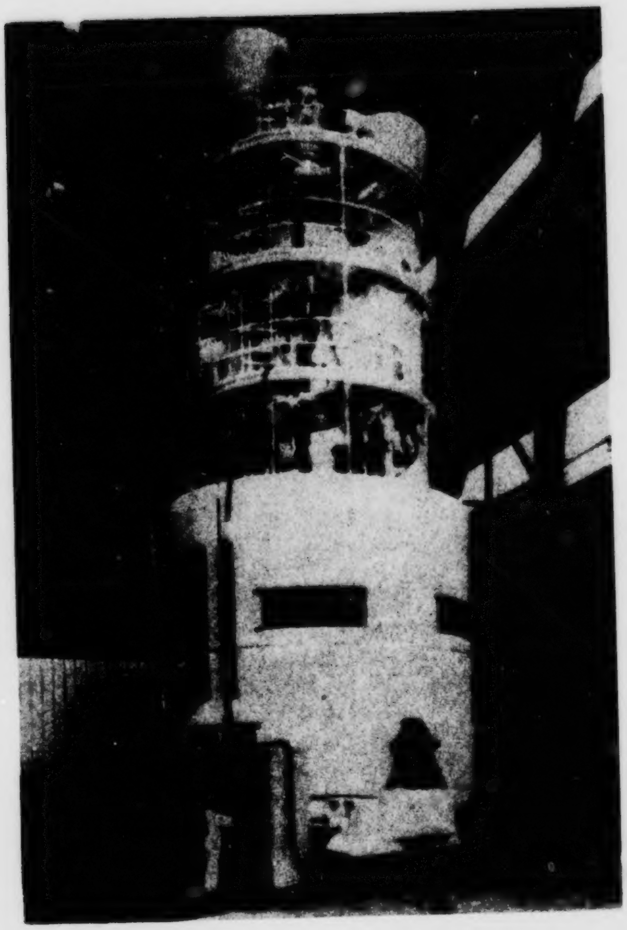


Fig. 4-36
Cutterhead for shaft drilling unit
(Courtesy Hughes)

Fig. 4-37
Blind shaft boring machine
(Courtesy Robbins)



.3 Blind Shaft Drilling Units

With this method, the drilling unit remains at the top of the shaft and transmits power to the cuttinghead through a rotating drill stem. This system has been used for over 20 years for small diameter shafts (less than 2 m), but recent technological improvements, particularly in stabilization and cuttings removal have extended the range of application of the method to shafts up to 5 m in diameter and up to 700 m deep. Cuttings are removed by reverse circulation systems using air or water. (Figs. 4-35 & 4-36)

Typical results for a 3.1 m diameter shaft, 680 m deep are a drilling time of 128 days, and 30 days to case and grout the lining, speeds 4 to 6 times conventional methods.

.4 Blind Shaft Borers

These machines are the direct application of tunnel boring machines to vertical or inclined shafts. The major difference between the TBM and the Shaft Boring Machine (SBM) is in the muck handling arrangements. Scrapers on the cutterhead move the cuttings towards the centre of the face where they are collected by bucket elevators and delivered into hoppers in the upper part of the machine. From this point they are discharged into the standard sinking hoppit. (Fig. 4-37)

An alternative method of muck removal being developed in Germany, the 'wet' process, utilizes an hydraulic system. A 200 mm diameter suction pipe is located in the cutterhead and connected to a pump. The cutterhead is always submerged in water. Scrapers move the cuttings into the path of the suction pipe, and are removed by a one-stage pump to a feed hopper, and thence via a second pumping system to the surface. Cuttings up to 90 mm are within the capacity of the first-stage system.

Machines are currently available with capacities up to 9 m diameter and 900 m depth. Typical performances in a 7.5 m diameter shaft (final depth 350 m) have been a best advance of 5 m in a 24 hour day with boring rates of between 1.0 and 1.5 m/h with the 'dry' mucking process. A 5.1 m diameter shaft, 270 m deep was bored using the 'wet' mucking process at maximum rates of 3 m/d and 20.9 m/week.

Machine systems include guidance by lasers, ventilation, temporary support, and the ability to line the shaft independently of the boring operation.

SBM's have the advantage that access is not required to the shaft bottom, and they do not suffer from the potential deviation problems that shaft drills can face. However, it is still early days with this particular method and further developments in both the dry and wet muck handling systems are needed to improve the reliability and performance of the machine. It is possible that some combination of the wet and dry methods will provide the answer.

465 APPLICATIONS

There is no doubt that the speed with which new mines can be brought into production is very dependent on the rate at which the shafts can be sunk and the access drivages driven. Mechanization of these activities is a must from all viewpoints, whether it be economics, environment, manpower shortage, productivity or safety, if the mining industry is to continue to be able to develop mineral deposits.

Equipment and techniques for boring horizontal tunnels, and boring or raising inclined shafts and raises are now well proven and reliable and can be immediately applied to mining industry needs. Shaft drilling and boring methods are newer but have sufficient practical application experience behind them to justify serious consideration by the mining industry.

These areas are amongst the last to remain unmechanized as far as the mining industry is concerned. The time is now ripe for this state of affairs to be changed for the ultimate benefit of all.

Fig. 4-38
Mobile hydraulic
impactor type
roof sealer
(Courtesy Tamrock)

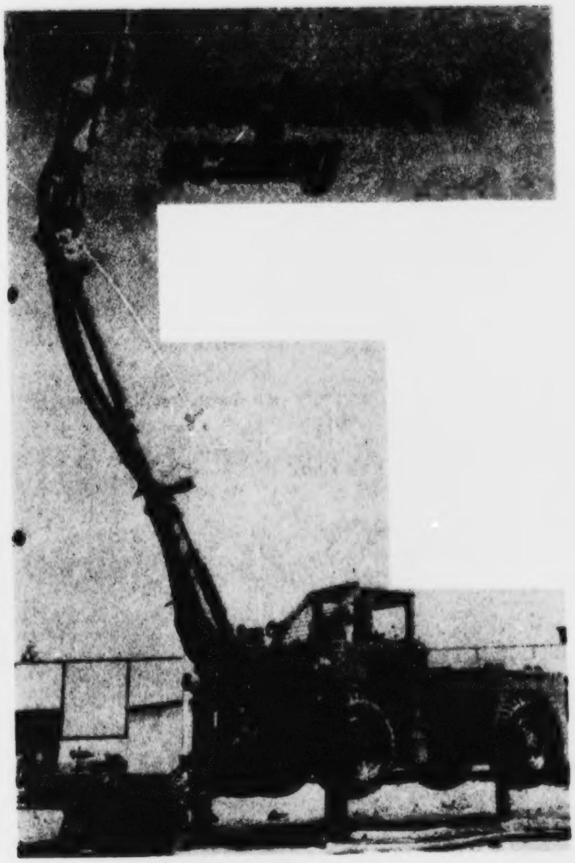


Fig. 4-39
Hydraulic impact breaker -
static type
(Courtesy Teledyne)

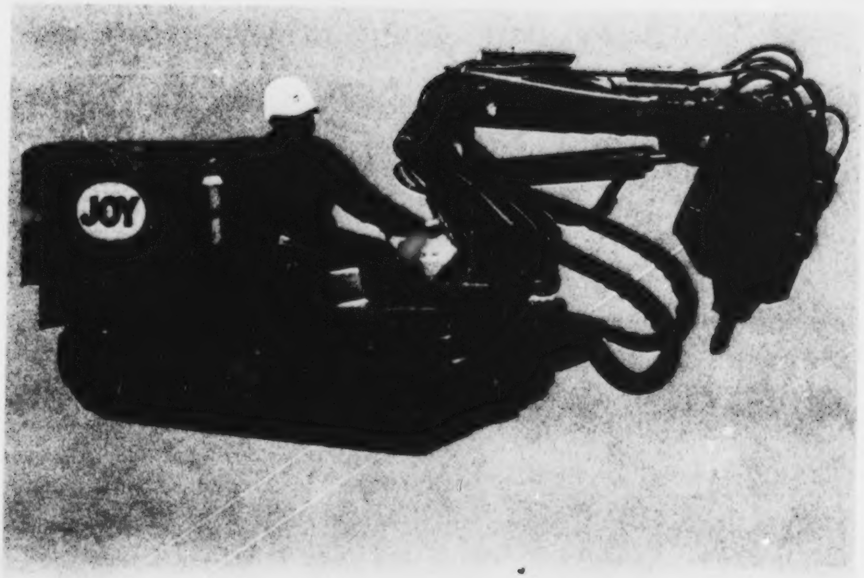


Fig. 4-40
Hydraulic impact breaker -
mobile type
(Courtesy Joy)

4.7 SAFETY AND CONTROL

These are areas where development is continually taking place. It is, therefore, almost impossible to note and record all these changes in an investigation of this type.

We have, therefore, limited the scope to major developments in equipment that have a major effect on the industry. Many of these, particularly in the control field are the result of the application of computers and microprocessors.

The developments that do or could make a significant contribution are:

Safety

- mechanized scaling of roofs;
- impact breakers for secondary breakage;
- autoloading of explosives;
- initiation of electric detonators; and
- shaft signalling systems.

Control

- remote control of equipment;
- automatic control of equipment; and
- monitoring of conveyors, strata, ventilation and environment.

471 SAFETY

.1 Mechanized Scaling of Roofs

Scaling, the removal of loose rock from the rock surface after blasting, has traditionally been carried out manually from a variety of working platforms depending on the height of the working area. The operation has many hazards, from falling rock, flying chips, dust, awkward working position, etc. Means for mechanizing this process have long been sought, but whilst there is now equipment available, some disagreement exists as to the correct principle to use. Essentially, there are two types, the hydraulically rotated tooth, and the impactor (hydraulic or pneumatic). (Fig. 4-38)

Scaling with the rotating tooth is more effective in the sedimentary formations, whilst the impactor is better for the more massive rock formations, especially in the electro-hydraulic form. In both cases, adequate lighting is essential.

Equipment is now available on both diesel and electrically powered bases, with effective boom lengths up to 30 m, giving a probable practical working height of approximately 20 m.

The advantages offered by the equipment include safer operating; faster, more productive and more effective scaling; better working environment. These can generally offset the additional capital cost of the equipment.

.2 Impact Breakers for Secondary Breakage

The traditional methods of handhammer or explosives for carrying out the breaking of rock lumps at grizzlies and draw-points can now be eliminated along with their attendant hazards.

The development of the hydraulic and pneumatic impact breakers, which revolutionized this field in the construction industry, is now providing the mining industry with a much needed tool.

Available as either a static unit or as a mobile trackless unit, fitted with hydraulic or pneumatic impact units, they have been making significant contributions, since the first prototypes appeared in the mid-70's, to improving productivity, and safety, and eliminating environmental problems underground. Production increases from drawpoints have been as much as 50% higher, downtime in crushers due to blockages have been reduced, and grizzly accidents almost eliminated. (Figs. 4-39 & 4-40)

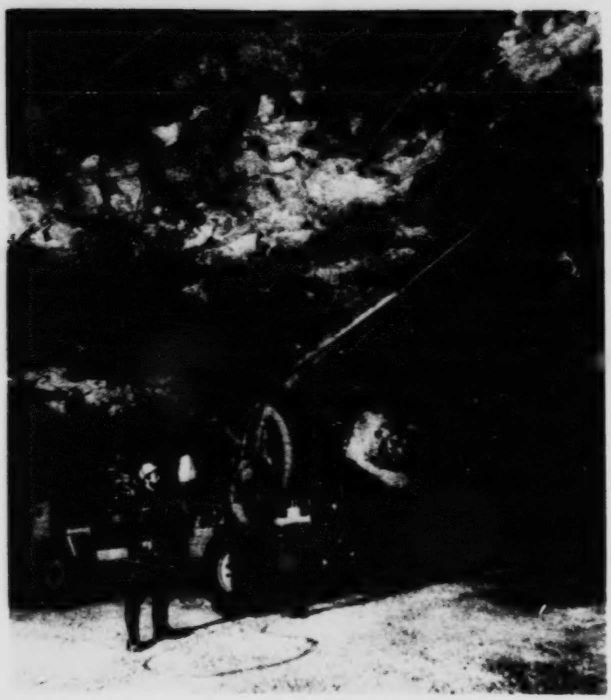
Units are available with blow energies ranging from 135 to 2750 kg/m, enough to meet any requirement.

.3 Autoloading of Explosives

Fully mechanized underground charging has long been a major objective for blasting personnel. Specialized vehicles, particularly for handling ANFO have been available for several years. These mount powder vessels and storage facilities, with a boom or scissors mounted working platform, from which the operator manually inserts the charging hose. These systems have the disadvantages of limited coverage from one vehicle position and problems of hose insertion where long holes have to be charged.

A recent improvement now enables holes up to 50 m long over an area up to 10 m wide to be charged by remote control. The unit has a telescopic boom which carries a flexible and easily operated trunk. This charging robot can be remotely controlled by one man, and is equipped with a hole depth gauge which makes it possible to detect caved-in holes and to improve the distribution of the explosive throughout the hole. The unit also provides the facility for the mechanical insertion of the detonator primer. (Fig. 4-41)

Fig. 4-41
Mobile remote charging unit
(Courtesy Nitro Nobel)



The unit offers the facility for safer charging, especially in the vicinity of open rooms, faster and more effective charging and reduced manpower requirements.

.4 Initiation of Electric Detonators

Traditionally, the method of coupling electric detonators has been to join each wire manually to obtain a complete circuit. This system is fraught with hazards--missed detonators, poor connections, difficulties in obtaining a circuit in wet conditions, liability to pre-ignition due to stray electric currents, large rounds having high resistances and requiring special means of detonation.

A new system has been developed, called Magnadet, which eliminates most of these difficulties. The wires from the detonator are attached to a ferrite toroid in such a manner that they form the secondary winding of a transformer. This connection is made during the manufacturing process. The toroid and the couplings are set within a plastic sheath to prevent mechanical damage, and they are colour coded and stamped with the delay number for identification.

To couple up a round, the shotfirer only has to pass a length of plastic covered copper wire through the centre of each toroid (thus forming the primary circuit) and connect the two ends of the wire to the exploder.

Connecting the circuit and checking the round is thus a relatively simple business. Savings of up to 20% on charging and firing times have been achieved with this new system. Rounds of up to 200 shots have been successfully fired, as have rounds completely submerged in water.

The initiation of the detonator relies on the application of a very high frequency discharge, from the special exploder into the primary wire. This causes a current to be generated in the secondary windings in each toroid of sufficient value to ignite the detonator. Since the current flow is instantaneous, all the connected detonators are initiated at the same instant. The circuit resistance is solely that of the primary connecting wire and is independent of the number of shots to be fired.

The magnadet system cannot be initiated by radio communication frequencies; thus, in these times of modern communications underground, it appears to offer a safe, reliable, almost foolproof method for the electrical initiation of explosives.

.5 Shaft Signalling Systems

In the past, shaft signalling systems have relied -- and mostly still do today -- on the operator ringing a predetermined number of bells according to a communication code. This system presupposes that the cage or skip attendant rings the correct code, and that the hoist operator receives and correctly interprets the signal. Judging by the speed with which these signals are often made, the margin for error is large.

New electronic systems are now in operation whereby the operator just pushes a button that corresponds to the particular signal he wishes to send. This produces a signal of constant clarity and duration irrespective of the operator using it. It is quicker and easier to use, and more important, the risk of human error and possible cage accidents is reduced. The system produces an accurate interpretation of a given signal; thus to transmit the signal 4-2, the operator simply pushes two buttons, the four and the two. It is not possible to stab the same button twice in quick succession--a built-in delay prevents such an accidental signal.

Additional features available are visual signals, and a microprocessor recording unit and printer which reports and records all signals sent as well as the time they were sent.

This development should be of great interest to mining companies, particularly from the safety aspect.

472 CONTROL

The advent of the computer and the microprocessor has revolutionized the mining industry. The potential exists to extend considerably the application of remote and automatic control, and to monitor operations throughout the mine.

Reference has already been made and comments given, in Sections 4.3.3.3 and 4.4.3.1, on the applications of remote control to LHD vehicles and of automatic control of trains and drill jumbos underground.

One of the most common underground applications for remote control techniques is the control of conveyors. It is now possible for a single operator in a central station to control the operation of a complete conveyor system including sequential starting and stopping, with observation of transfer points by remote TV.

This degree of control, however, has now been extended to automatic operation. The application of microelectronics not only enables automatic control to be applied, but also provides the means for monitoring the operations, the equipment, and the environment.

Automatic control of conveyors, trains, hoisting, bunkers and pumping operations is becoming common place. The systems also monitor the mine ventilation-flow, pressure, analyse the air for gasses (CO, CO_2, NO_x, CH_4, etc.), and can make adjustments to compensate for changes. The recent increases in the use of diesel, the need in most Canadian mines to heat the mine air, and clean it of exhaust gases, and the impetus to reduce ventilation costs have virtually made ventilation monitoring and control a necessity. One copper-zinc mine has achieved dramatic savings by using a mine ventilation, monitoring and control system. Use of this system has cut ventilation energy requirements by 25% through more accurate control of the airflow; air heating energy has been cut by 45%, saving 260 000 L of heating oil; and the wet bulb intake temperature has been lowered from 5°C to 1°C, thus improving the working conditions underground.

Ground movements and stresses can be monitored to (a) provide data for better understanding of the insitu state of the rock strata and (b) to give early warning of the onset of hazardous ground conditions. These systems should enable better ground control to be achieved and hence more productive and safer mining.

The availability of control and monitoring systems offer the mining industry the opportunity manage the mining operations more effectively and safely. The costs of the systems are more than offset by the gains made in productivity (increases of over 20%), reduced manpower requirements, increased safety, and better planning based on factual information.

We believe that there is a tremendous scope for the application of these systems to the Canadian mining industry. The variety and scope of the available systems is such that it is possible to control or monitor almost every operation within the mine.

TABLE 4-1

Comparison of Hydraulic & Pneumatic Drilling
(ratio)

	Hydraulic	Pneumatic
Capital Cost	1.5 - 1.9	1
Production per capital dollar	1.2 - 1.1	1
Operating Cost/tonne	0.5 - 0.7	1
- Labour/tonne	0.5 - 0.6	1
- Steels & Bits/m drilled	0.4 - 0.9	1
- Power	0.4 - 0.6	1
- per m drilled	0.20 - 0.35	1
- maintenance/tonne	0.5 - 1.7	1
Penetration rate	1.5 - 2.0	1
Tonnes/hole drilled	1.2	1
Noise levels (dBA)	0.8 - 0.85	1

TABLE 4-2

Comparison of ITH Drilling with Longhole Drilling (HTD) and Rotary Drilling (LRD)

	ITH	LHD	LRD
Capital Cost	1	NA	2.2
Cost/tonne (operating)	1	2.5	3.1
Tonnes/m	1	0.13	1.3
Cost/m drilled	1	0.3	4.2
Cost/m - labour	1	0.4	3.1
- bits	1	0.2	8.0
- maintenance	1	0.4	4.5
- misc. supplies	1	0.5	NA
Blasting cost/tonne	1	2.25	NA

Note: Costs per tonne considerably better because of improved production per metre drilled.

TABLE 4-3

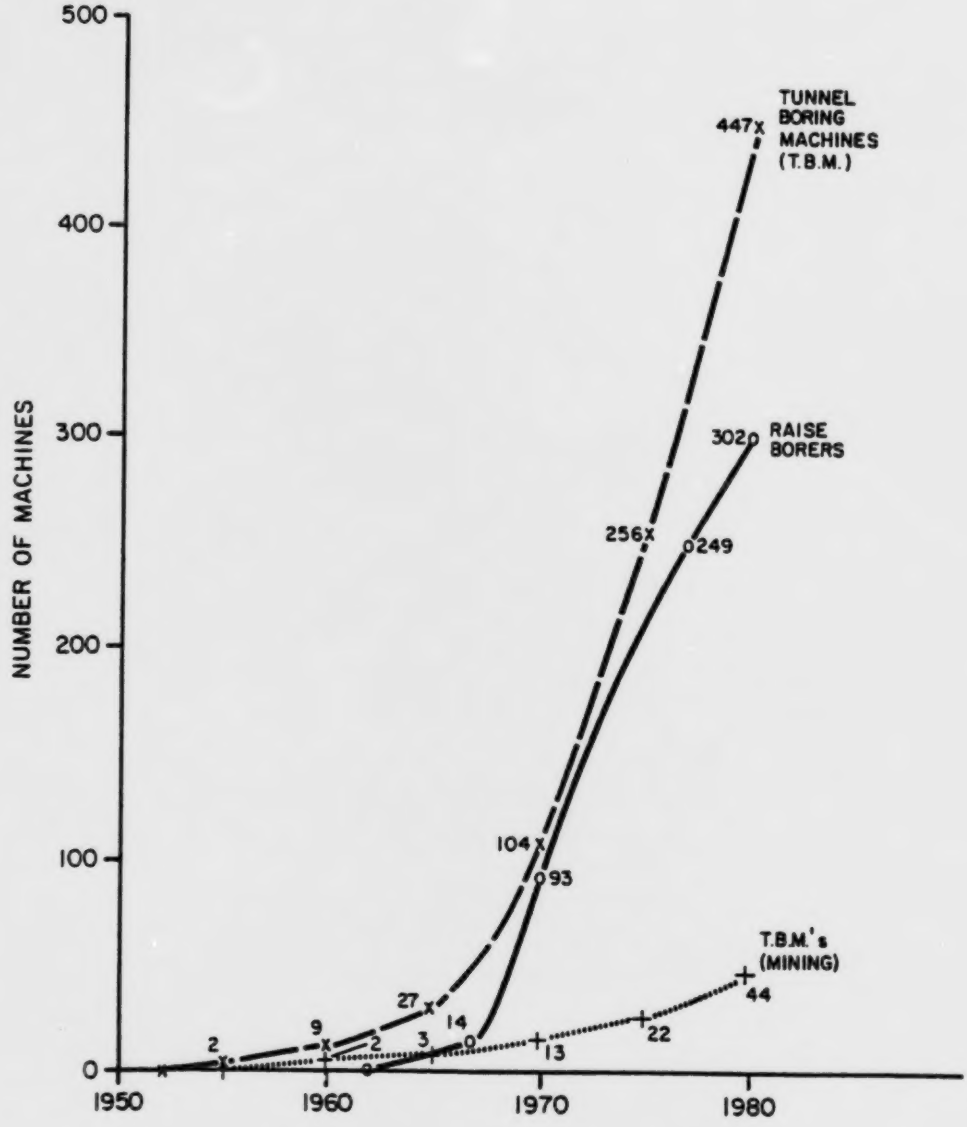
Main Haulage Systems in Canadian Metal Mines

	No. of Mines			
	(25)		(48)	
	1974		1979	
	No.	%	No.	%
Locomotive & Mine Cars				
- Battery	10	35	19	35
- Trolley	9	30	15	27
- Diesel	2	7	5	9
Trucks - Diesel	2	7	11	20
Load/Haul/Dump - Diesel	2	7	1	2
Belt Conveyors	4	14	4	7

EXHIBIT 4-4

NUMBERS OF PROJECTS (CUMULATIVE)

(TUNNEL & RAISE BORING MACHINES)



PART 5

CASE HISTORIES

PART 5 - CASE HISTORIES

The following technical papers can be referred to for a more detailed description of the major developments discussed in this paper.

A. DRILLING

1. Application of Electro-Hydraulic Drilling at Falconbridge Nickel Mines Ltd.
   J.N. Szuck; J.A. Riddell and K.H. Singh - CIM April 1981

2. Computer Control Comes to Hard Rock Drilling
   K. Dahl - Tunnels & Tunnelling May 1981

3. Down the Hole Drilling at Geco
   R.N. Weeks - CIM February 1981

4. Mechanization in Lateral Development
   J.R. Wright - CIM February 1979

B. LOADING

1. Remote Control Load-Haul-Dump Units
   T.L. Lakanen - Mining Congress Journal March 1980

2. Major Technological Advances in the Design of Trackless Mining Equipment in the '70s
   R.A. Richard - CIM April 1980

3. Development of Electrical Equipment at Sherritt Gordon's Fox Mine
   E.N. DuRussel and J.L. Muirhead - E & M.J. August 1981

4. Construction of the Meggett Water Tunnel
   A. Young - Tunnels & Tunnelling April 1981

5. Track Drifting with the Hagglung System
   G. O'Gorman - CIM February 1981

C. TRANSPORTATION

1. Computer Controlled Ore Transportation at the Kiirunavaara Iron Ore Mine in Kiruna, Sweden
   ASEA Pamplet AS20-114E Ed. 1 - November 1980

2. Low Profile Trucks Keep Underground Costs Down
D. Martin - Tunnels & Tunnelling July 1981

3. U.S.B.M. Develops Portable Crusher for Underground
Hard Rock Mines - R.L. Schmidt - E&MJ. Jan Pgs. 72-75.

4. Feed Breakers at White Pine Copper
R.E. Niemela - E & M.J. April 1980 Pgs. 76 - 80

5. High Speed Belt Conveying
P.D.H. Comley - Mining Technology June 1981

6. Selby Conveyor - World's Most Powerful Conveyor
Anderson Strathclyde leaflet 52-020 April 1980

7. Booster Conveyors
Huwood leaflet HW/54a 1981

8. Angle Stations
Cable Belt leaflet B.11a. 1980

9. Muck Climbs 262 ft. Shaft
Engineering News Record March 1980

10. Air System Takes Rock to Surface
D. Brezovec - Coal Age March 1981

D. SUPPORT

1. Split-Sea Bolt - Ingersoll Rand leaflet 1978

2. Swellex Bolt - Atlas Copco leaflet 1981

3. Cementitious Rock Bolt Cartridges for Anchors
D. Martin - Tunnels & Tunnelling May 1981

E. ACCESS DRIVAGE

1. Application of Hard Rock Continuous Miners to Cut & Fill Stoping G.B. Sparks - Mining Congress Journal May 1980

2. Robbins Digs on in Europe and Australia
Water Power & Dam Construction February 1980

3. Operation of Wirth Tunnel Boring Units in Granite
W. Hildebrand publ. in The Technology and Potential of Tunnelling

4. Mining Applications of Tunnel Boring Machines
   H.J. Handewith - CIM November 1980

5. Application of Hard-Rock Boring Machines in Deep South African Gold Mines of Anglo American Corporation
   E. Schmid, J.G. Taylor - I.M.M. 1977

6. An Inclined Gallery through Hard Rock
   P. Hambach - Tunnels & Tunnelling June 1970

7. 20 Foot Diameter Blind Shaft Drilling
   J.P. Carone and D.A. Whitley - CIM June 1981

8. Blind Hole Raise Boring
   C.F. Smith - Mining Congress Journal June 1980

9. Shaft Sinking Today - Boring
   Business Tomorrow
   M. Grieves - SME-AIME January 1979

F. SAFETY

1. Impact Breakers Improve Productivity
   H.W. Seavey - Mining Congress Journal April 1981

2. Mine Wide Monitoring System at San Manuel
   C.A. Metcalf and K.L. Billhartz W.M.I.E.C.
   September 1981

APPLICATION OF ELECTRIC-HYDRAULIC DRILLING AT FALCONBRIDGE NICKEL MINES LIMITED

by

J.N. Szuck, J.A. Riddell and K.H. Singh

Falconbridge Nickel Mines Limited
Canadian Nickel Division - Sudbury Operations
Falconbridge, Ontario

Paper to be presented at the
83rd Annual General Meeting of
CIM - 1981

APPLICATION OF ELECTRIC-HYDRAULIC DRILLING AT FALCONBRIDGE NICKEL MINES LIMITED

ABSTRACT

Hydraulic rock drilling equipment has been in use at Falconbridge over the past two years in both production and development situations.

This paper deals with the experience gained in the application of hydraulic drilling in our mines and discusses the benefits and problems encountered during their use. The paper primarily focuses on technical, operating, maintenance, and economic aspects of the machines and does not attempt to compare the performances of different machines from the various manufacturers.

INTRODUCTION

The Sudbury Operations of Falconbridge Nickel Mines Limited are comprised of five operating mines and two mines presently under development. All the ore production comes from underground mines mostly at depths below 1000 meters. The Company, faced with increases in operating costs and difficult mining conditions, has continued to seek solutions with the help of new mining technology.

The development of electric-hydraulic drilling machines in Europe had been pursued with great interest and in 1979, a decision was made to test these machines to prove their technical and economic viability under our operating conditions. Since then, a number of electric-hydraulic machines have been evaluated in different situations and currently nine hydraulic machines are being used in our various mines. These machines are comprised of 2-boom drilling jumbos for development, one-boom machines for longhole drilling and a large hole diameter drilling rig for blasthole drilling. One hydraulic jumbo has also been used in a cut-and-fill stope.

The discussion in the paper primarily focusses on the application of hydraulic machines rather than on the features of the various drills. An attempt has been made to concentrate on Falconbridge's experience gained during the use of various machines. This experience has been presented in a case history form and incorporates economic, operating and maintenance aspects of the hydraulic drilling equipment used in Strathcona, East and Fraser Mines. The paper includes a review of bits and steel problems encountered during drilling with the hydraulic equipment.

BACKGROUND INFORMATION ON HYDRAULIC DRILLING EQUIPMENT

Before discussing the results of hydraulic drilling, it would be useful to give some background information on the various machines tested at Falconbridge. Tables 1 and 2 summarize the main features of these machines. Table 3 gives the statistical operating data for two-boom jumbos.

TABLE 1

TWO-BOOM JUMBO SPECIFICATIONS

	UNIT A	UNIT B	UNIT C
Carriage Description	4 wheel drive, articulated steering.	4 wheel drive, articulated steering.	Rigid frame, 4 wheel drive
Booms	telescopic parallel holding.	non-telescopic booms rotate.	telescopic feed, parallel holding.
Face coverage (WxH)	6.1 m x 4.6 m	7.0 m x 5.0 m	9.6 m x 5.7 m
Length	11.27 m	11.27 m	11.27 m
Width	1.85 m	1.83 m	2.36 m
Height	2.69 m	2.49 m	2.44 m
Drill Hole Depth	3.66 m	3.66 m	3.66 m
Hole Size	35 mm - 41 mm	32 mm - 41 mm	45 mm - 64 mm
Steel Size (Hex)	32 mm	32 mm	32 mm

TABLE 2

LONGHOLE DRILLING EQUIPMENT SPECIFICATIONS

	UNIT D	UNIT E
Length	3.63 m	3.89 m
Width	1.35 m	1.83 m
Height	1.93 m	1.57 m
Hole Size	45 mm to 64 mm	51 mm to 115 mm
Steel Size	32 mm Rnd 1.52 m long	32 mm Rnd 1.52 m long
		51 mm Rnd 1.52 m long
Remarks	power pack is separate & mounted on skids	-

TABLE 3

TWO-BOOM JUMBO - STATISTICAL OPERATING DATA

Unit	Working Hours	Oper. Hours	Mech. Hours	Available Hours	% Avail.	% Mech. Avail.	% Utiliz.	Meters Drilled
A	2,760	552	322	2,438	88.3	63.2	22.6	27,209
B	2,436	859	380	2,056	84.4	69.3	41.8	42,789
C	3,285	1,361	749	2,536	77.2	64.5	53.6	64,553

DEFINITION OF TERMS

$$\text{Availability} = \frac{\text{Working Hours - Mecnanical Hours}}{\text{Working Hours}}$$

$$\text{Mechanical Availability} = \frac{\text{Operating Hours}}{\text{Operating Hours + Mechanical Hours}}$$

$$\text{Utilization} = \frac{\text{Operating Hours}}{\text{Working Hours - Mechanical Hours}}$$

HYDRAULIC DRILLING AT STRATHCONA MINE

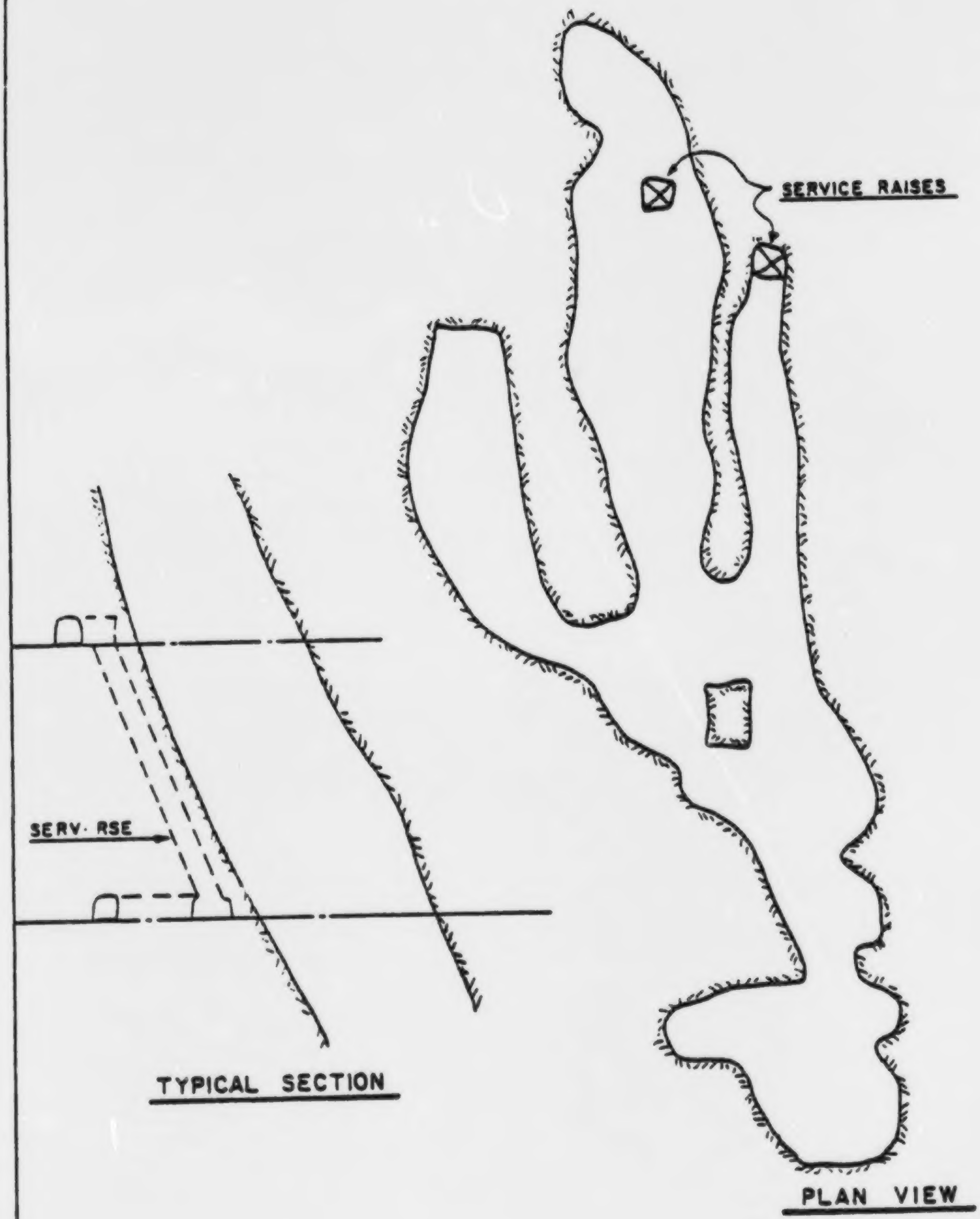
Strathcona is the major producer of the Company. Primarily using cut-and-fill stoping methods, the total annual production is 2 million tonnes. About 60% of the ore reserves have been extracted and general mining conditions are becoming more and more difficult with time.

The application of hydraulic drilling was considered for two purposes: firstly, to evaluate the feasibility of hydraulic drilling concepts in a captive Load Haul Dump (L.H.D.) cut-and-fill stope; and secondly, to use longhole drilling for pre-pinning stope backs with grouted cable bolts. The main objective here was to expedite the stoping cycle.

HYDRAULIC DRILLING JUMBO IN A STOPE

In the fall of 1978, an electric-hydraulic jumbo (Unit A) was placed in a cut-and-fill stope of the mine. This machine was obtained for test purposes only. This stope uses tight cut-and-fill breasting with jacklegs, stopers and a 4-yard scooptram. The stope was 122 m long and varied from 12 m to 46 m in width (see figure 1). In order to increase utilization of the jumbo, multiple working faces were scheduled. The machine was kept over a one year period and satisfactory results were achieved. The test demonstrated the following benefits:

1. Drilling Rate: The penetration rate was 1.43 m/min as compared to 0.46 m/min for jacklegs. The average drilling rate worked out to 0.82 m/min which included changing bits and steel, collaring, setting up, etc., as compared to 0.21 m/min for jacklegs. As a result, the 12.8 x 3.7 m breast required 3 hours drilling with one man while the same work using jacklegs required 2 men for a complete shift.


SERVICE RAISES
SERV. RSE

TYPICAL SECTION

PLAN VIEW

MECHANIZED CUT & FILL STOPE
WITH HYDRAULIC DRILL JUMBO

FIG: 1

2. Environmental Conditions: The lack of fog at the working face allowed the operator to see clearly what he was doing. The noise level at the operator's position with both drills operating was 107-108 db, compared to 110 db or higher with muffled jacklegs.

3. Operator Acceptance: The jumbo was well accepted from the time it was introduced in the stope, mainly because the physcial effort required was greatly reduced.

4. Mechanical Availability: The short term results of this test were encouraging and a mechanical availability of 63.2% was achieved. Since the machine utilization in the stope was low, 22.6%, this low mechanical availability did not affect the drilling operation. A large portion of the maintenance work was performed by the supplier's service department. As a result, we did not obtain absolute maintenance costs, but the figures attained indicated they were lower than current pneumatic drills used in the stope by approximately 50%.

The unit cost of the operation (including capital costs) indicated that hydraulic drilling costs were twice as much as a jackleg drilling on a per unit drilled basis. Since the nyraulic drill jumbo is part of the total stoping equipment, it is necessary to consider not only the unit operation of the hydraulic drill itself, but the impact on the total stoping operation. Performance data summarized in Table 4 indicated that hydraulic drilling proved to be successful in a captive L.H.D. cut-and-fill stope. Direct stoping costs were reduced by 16%.

TABLE 4

PERFORMANCE COMPARISON

Basic Equipment	Hydraulic Jumbo & 4 Yd. L.H.D.	Jacklegs & 4 Yd. L.H.D.
Ore Removed/Day Tonnes	444	292
/Month "	9,316	6,134
/Year "	105,223	62,877
Direct Cost $/tonne	.84y	y
Productivity Tonnes/M.S.	48.5	31.0
Utilization	22.6%	-
Penetration Rate M/Min	1.43	0.46
Average Drilling Rate M/Min (includes set up, bit changes)	0.82	0.21

The much higher hydraulic drilling rate resulted in a shorter overall time to mine a complete cut. Consequently, the ground conditions improved markedly over the course of the one year test.

In order to evaluate and compare further the actual performance, as well as to simulate various combinations of drill equipment, available faces and crew size, a computerized simulation model was used.

In a cut-and-fill stope, the unit operations consist of drilling (D), blasting (B), scaling and bolting (SB), mucking (M) and filling (F). The total cycle time (TT) in hours to complete a breast is:

$$TT = \{D/K + B + SB/n + M + F\}\ PA/Eff \text{ for one working face.}$$

$$= \min_{i = 1, \ldots, n} \{\max\ (D/K + B + F,\ SB/n-K + M + F)\}\ /PA/Eff \text{ for two working faces}$$

$$= \min_{i = 2, \ldots, n} \{\max\ (D/K + B + F,\ SB/n-K-1 + F,\ M + F)\} \text{ for three working faces.}$$

n: number of miners in stope

K: 1 for hydraulic jumbo = n-i for jacklegs

PA: The physical availability of all mining machines in the stope;
The "realistic" figure is 75%.

EFF: efficiency of all unit operations in the stope. An average of 60% is assumed for our "regular" stope.

Table 5 shows the results of the calculations based on this mathematical model. When conditions are similar, these results compare closely to that of the actual performance in the stope, as given in Table 4.

TABLE 5

COMPUTER MODEL RESULTS OF THE STOPE PERFORMANCE

	Productivity tonnes/man shift	Production Rate tonnes/month	Direct Stoping Cost $/tonne	H.D. Utilization %
A. Hydraulic Drill				
a) One face				
i) two men	43.3	4,082	.78 x	14.4
b) Two faces				
i) two men	49.4	4,236	.76 x	14.9
ii) three men	44.0	5,647	.81 x	20.0
iii) four men	37.3	6,383	.88 x	22.6
c) Three faces				
i) three men	49.4	6,349	.76 x	22.5
*ii) four men	49.9	8,536	.75 x	30.3
iii) five men	39.9	8,536	.85 x	30.3
B. Jackleg or Longtom				
a) One face				
i) two men	39.0	3,343	.86 x	
b) Two faces				
i) two men	45.8	3,924	.79 x	
ii) three men	33.7	4,335	.93 x	
iii) four men	33.0	5,647	.94 x	
c) Three faces				
i) four men	42.6	6,318	.88 x	
*ii) five men	29.8	6,372	x	
iii) six men	28.8	7,381	1.02 x	

*Compare with Table 4 for our actual results where same number of miners were employed in the stope.

NOTE: A 5 yard LHD unit is used in all instances.

From the results obtained (Table 4), it can be concluded that the hydraulic drilling jumbo appeared to be economically attractive. The question, however, still remained as to whether the high capital cost of such a machine could be justified in a production stope. For this purpose, an economic analysis was carried out to calculate the net present value (NPV), internal rate of return (IRR) and the payback period for two different options (see Appendix 1).

In the first option, the results of the hydraulic drilling jumbo were compared with the longtom jumbo without any increase of manpower. This resulted in an increase of ore production from 6,300 tonnes to 9,100 tonnes with the hydraulic jumbo. The second option was to increase manpower for a longtom jumbo so that the stope production could be the same as for the hydraulic jumbo. Two additional men were required. The economic results are shown in Table 6.

TABLE 6

ECONOMIC COMPARISON BETWEEN HYDRAULIC AND LONGTOM JUMBO STOPE

	Hydraulic Drill Stope		Longtom Stope	
	Option I	Option II	Option I	Option II
NPV	$ 5.6 M	$-1.7 M	$ 4.9 M	$-1.8 M
IRR	130%	156%	-	-

NOTE: Option I, the annual cash flow was based on increased revenues from ore production. Option II, the annual cash flow was based on operating revenues.

It is evident from these results that the investment in a hydraulic jumbo could be justified in a cut-and-fill stope.

However, it is more difficult to quantify the other benefits (beyond better ground conditions) which accrued from the faster mining cycle. Nevertheless, there must be definite savings in that, for a given rate of production, fewer stopes would be required, hence less development work would be needed to establish those stopes initially and to open up replacement work places later.

HYDRAULIC MACHINE FOR DRILLING 51 MM DIAMETER UPHOLES

An electric hydraulic longhole machine was obtained in 1979, on a two month rental basis, for drilling upholes. The results indicated that the number of units could be doubled over a wagon drill equipped with pneumatic drills, but the machine was too large for the narrow work places that required cable bolting.

In mid-1980, another hydraulic drill was tested for uphole drilling. Several 30 m, 51 mm diameter upholes were drilled successfully but the carrier was also large.

The supplier of that drill had a considerably modified model coming onto the market with a carrier that would fit inside our shaft cage and a separate power pack on skids. Before making a commitment to purchase this machine, an economic analysis was carried out for hydraulic and pneumatic drills. It was assumed that a hydraulic drill with only one operator could drill 75 m/shift compared to a pneumatic drill's 61 m/shift with two operators. This analysis showed an IRR for the hydraulic drill of 80% with a payback period of 15 months. Based on this conclusion, the modified model as described for Unit D was purchased. Specifications for this machine are given in Table 2, and figures 2 and 3 illustrate some of the features. There are two such machines on site drilling 17 m upholes for cable bolting in stope backs. Current drill

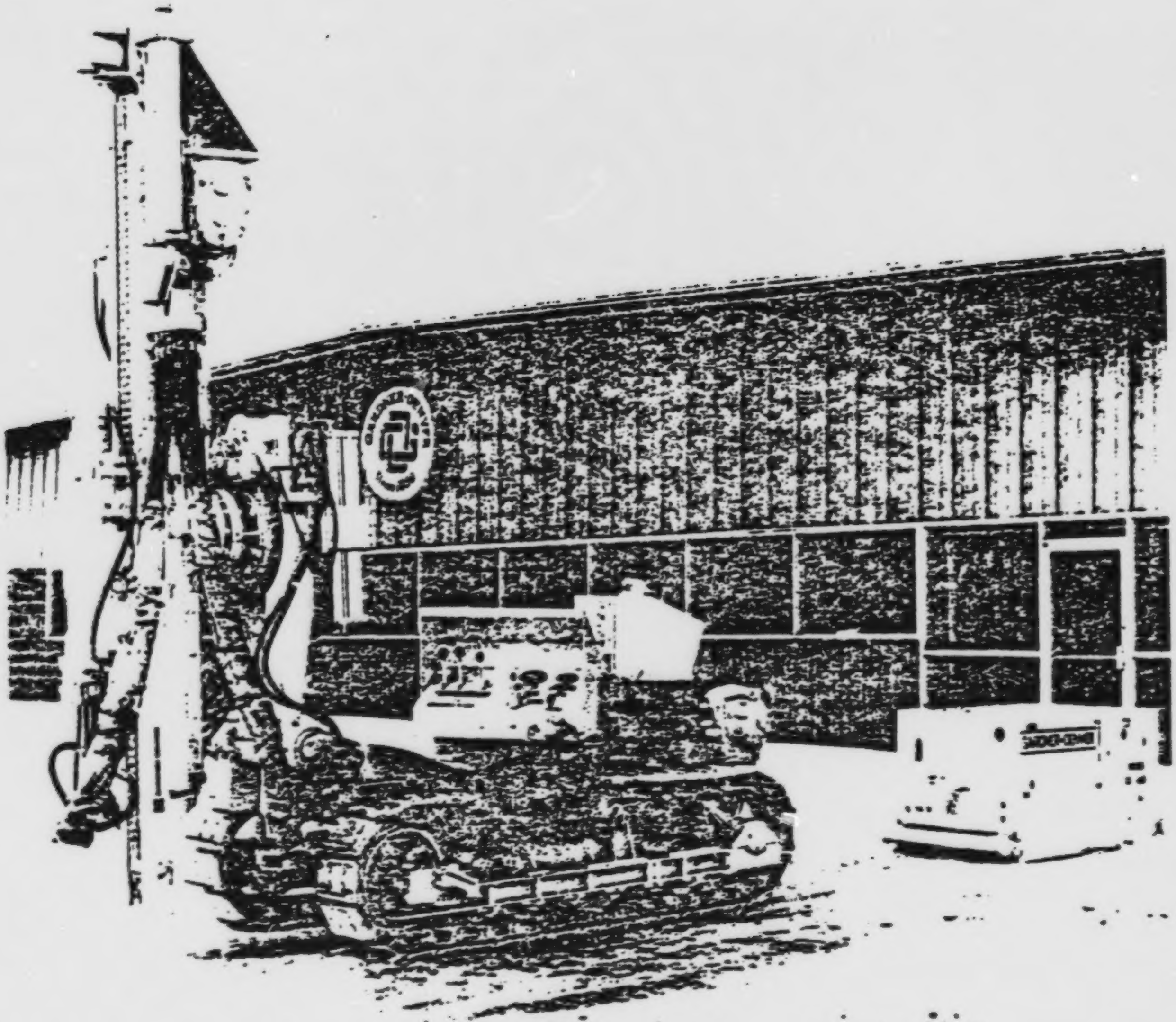


FIGURE 2 - HYDRAULIC LONG HOLE MACHINE

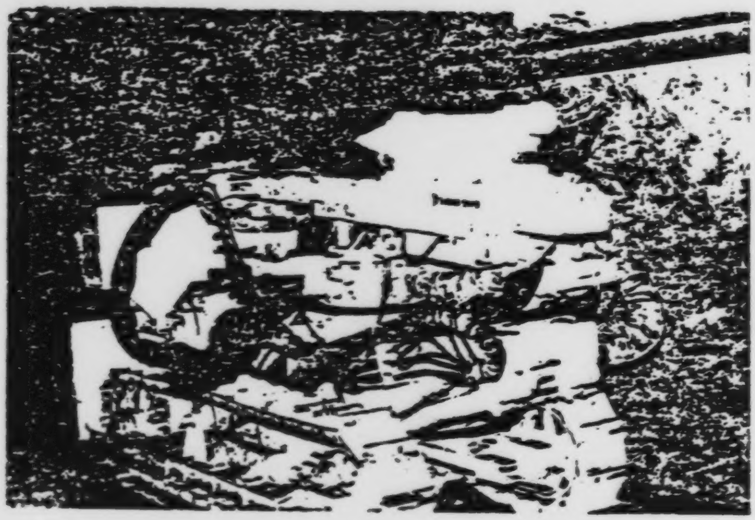


FIGURE 3 - ELECTRIC-HYDRAULIC LONGHOLE MACHINE

performance is 70 m/manshift compared to present pneumatic performance of only 25 m/manshift. This is within 92% of the forecast rate and still provides an acceptable internal rate of return. This increase in the drilling rate has permitted Falconbridge to cable bolt stopes in a shorter period of time. These stopes are being brought back into production much sooner thus helping maintain production rates.

The maintenance costs for this uphole drilling has shown that the hydraulic drill costs are approximately 10% less than pneumatic drills. These maintenance costs are compared with new equipment in both cases.

HYDRAULIC DRILLING AT EAST MINE

This is one of the older mines which has been re-developed from a cut-and-fill mine to bulk mining methods (^{1}). In order to achieve optimum economic results from this mine, planning was based upon the use of the most advanced drilling equipment. Two types of hydraulic machines were evaluated: two-boom hydraulic drilling jumbo for increasing pre-production drifting; and a hydraulic tracked drill for blasthole drilling.

HYDRAULIC DRILL JUMBO FOR DEVELOPMENT

During the later part of 1978, a study was conducted on three types of drill rigs for drift and stope development work. The three types compared were: a hydraulic drill jumbo, a pneumatic drill jumbo, and a longtom.

Although the capital cost for the hydraulic drill was higher, the study revealed it would have the lowest operating costs per unit of advance. In addition, the capacity of the hydraulic drill jumbo would allow completion of the development work in a much shorter time span. This study had also indicated that with a two-boom hydraulic drill jumbo, 127 m of advance in a 4.3 m x 4.0 m drift could be achieved per month versus an average of 75 m per month using a pneumatic drill jumbo. Experience has shown that an average of 106 m of advance per month was achieved using a two-boom hydraulic drill jumbo. Specifications and statistical operating data of this machine (Unit B) are shown in Tables 1 and 3.

The maintenance cost (per unit drilled) in the original study were estimated to be 20% higher than for a pneumatic jumbo, but our actual experience has shown that hydraulic maintenance costs are 50% less than pneumatic maintenance costs. However, it should be noted that most of our pneumatic drill jumbos are much older than the hydraulic drill jumbos.

From this experience, we have determined that our internal rate of return ranges from 45% to 80%, depending upon annual advances achieved. These results are shown in Table 7.

TABLE 7

DRIFT JUMBO COMPARISON

Meters/Yr	Shifts Required	Method	IRR on the Additional Capital Investment of $250,000
732	286	Pneumatic	
732	219	Hydraulic	45%
1158	453	Pneumatic	
1158	346	Hydraulic	70%
1219	477	Pneumatic	
1219	365	Hydraulic	75%
1273	498	Pneumatic	
1273	381	Hydraulic	80%

HYDRAULIC DRILL FOR LARGE HOLE DRILLING

Early in 1979, drilling suppliers were approached concerning the feasibility of drilling large diameter downholes of about 30 to 40 m in depth, using a hydraulic drill. As a result of this inquiry, a surface drill rig capable of drilling 115 mm diameter downholes was obtained on a test basis.

During the first few weeks of drilling, several problems were encountered. The worst was the large amounts of flush water required which resulted in the mine going dry when the machine was drilling. This problem was solved by adding a reservoir to re-circulate some of the water and a booster water pump to aid in flushing.

Attempts were also made to drill as deep as 46 m but this proved to be impossible. The unit operated very efficiently for downholes as deep as 21 m and then its penetration rate dropped off. It was found that the practical limit was 37 m.

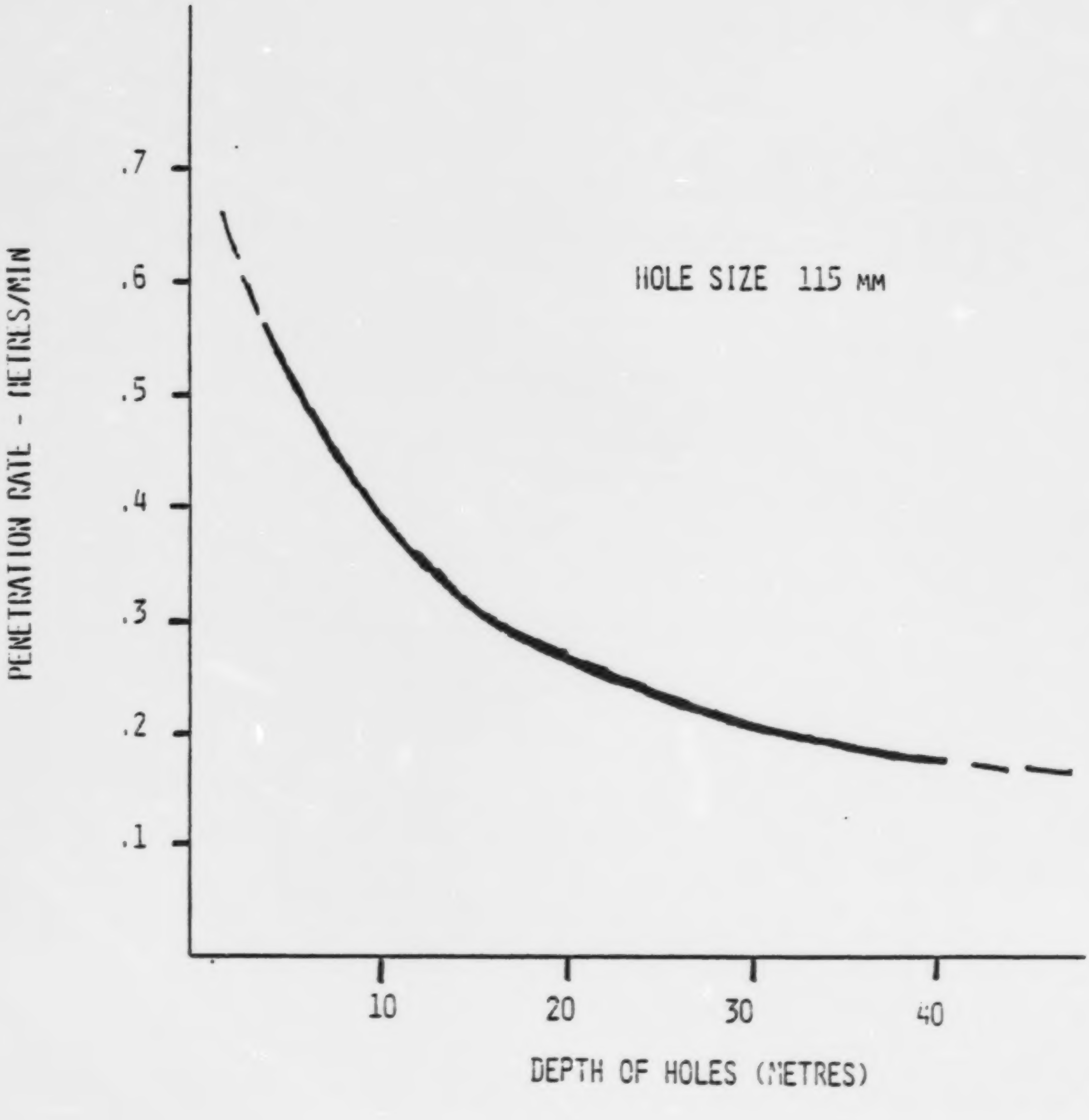
This test revealed that an average of 46 m/shift could be drilled vs 30 m/shift using a pneumatic in-the-hole (ITH) machine on high pressure air. The average penetration rate for various depths of holes are shown in figure 4.

The economic analysis showed that the additional expense in purchasing such a machine gives an IRR of 87% vs an ITH machine with a booster compressor. The maintenance costs, comparing new machines of each type, were found to be approximately identical.

The tests indicated that the hydraulic drilling of 115 mm holes could be economic for blastholes up to 21 m in length. This would be suitable for those orebodies where geological features or other design considerations limit the height of the blasthole stopes.

FIGURE 4

LARGE HOLE DIAMETER DRILLING PENETRATION RATES WITH A HYDRAULIC DRILL



After initial testing, a modified version of the prototype was purchased (see Unit E, Table 2). At present, this unit is being used temporarily for drilling 51 mm diameter upholes which are required for cable bolting.

HYDRAULIC DRILLING AT FRASER MINE

Fraser Mine, currently in pre-production stages, is slated to start production in the early part of 1982. With the pressure on a development program, it was considered important that the best possible equipment be used to maintain the development schedule. An initial investigation on the feasibility of selecting hydraulic drill jumbos for development was carried out. It showed that the additional investment in this type of equipment as compared to pneumatic would have an internal rate of return between 11% and 34% depending upon the various parameters considered.

It was estimated that a development advance rate of 150 m per month could be achieved in a 5 m x 4 m drift. Unit C (Tables 1 & 3) was introduced into this mine. Figures 5 to 9 show some of the features of this equipment.

After initial start-up problems were resolved, an average monthly advance rate of 133 m has been achieved in a multiple heading situation (figure 10). As the best monthly advance has been 182 m, the anticipated average rate of 150 m/month is clearly an attainable target. Table 8 indicates advance rates possible at Fraser based upon current performances in 3 to 4 headings. Assuming that there are 21 workings days/month, 498 shifts/year, the increase in advance with hydraulic equipment gives an IRR in the order of 80% as well as permitting a mine to come into production much earlier. This also improves the economics of the project by generating an earlier positive cash flow.

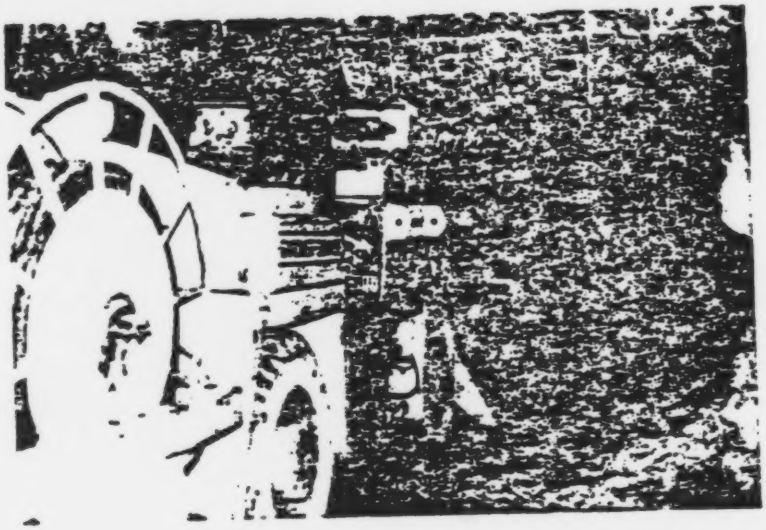


FIGURE 5 - HYDRAULIC DRILL JUMBO

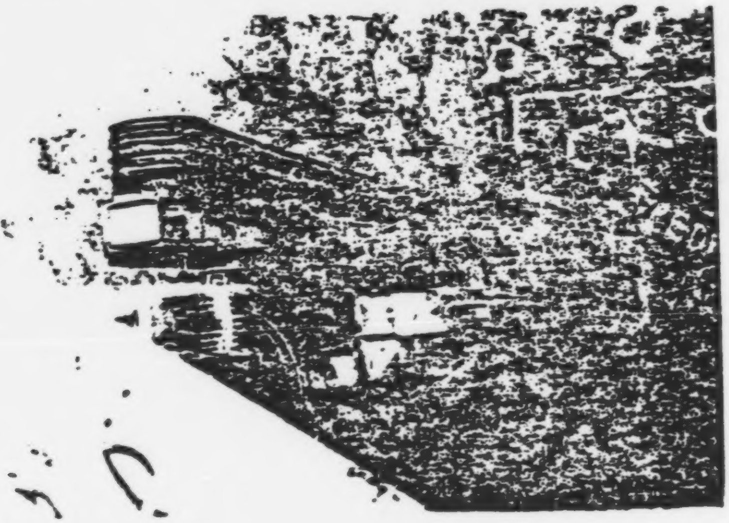


FIGURE 6 - HYDRAULIC DRILL JUMBO (Feed and Boom)

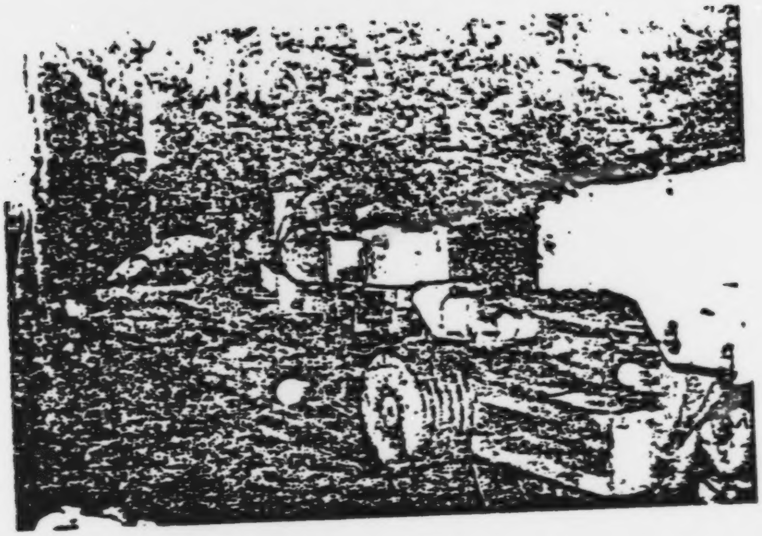


FIGURE 7 - HYDRAULIC DRILL JUMBO (Feed and Boom)

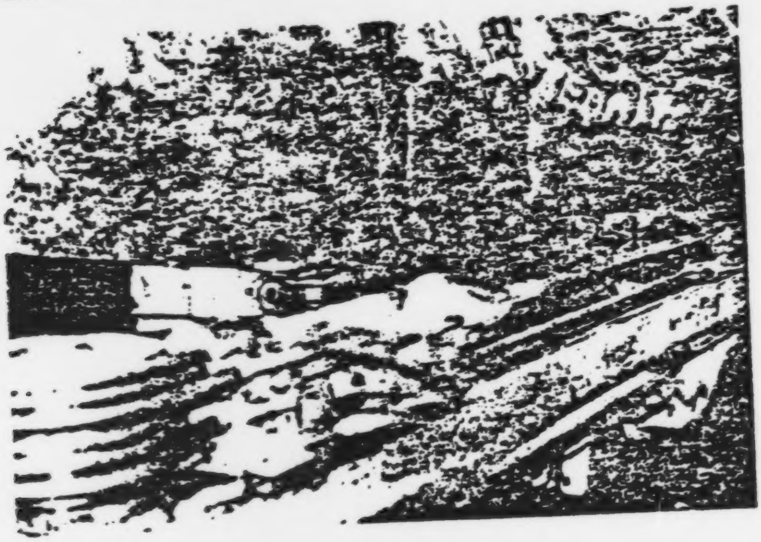


FIGURE 8 - HYDRAULIC DRILL JUMBO (Drill)

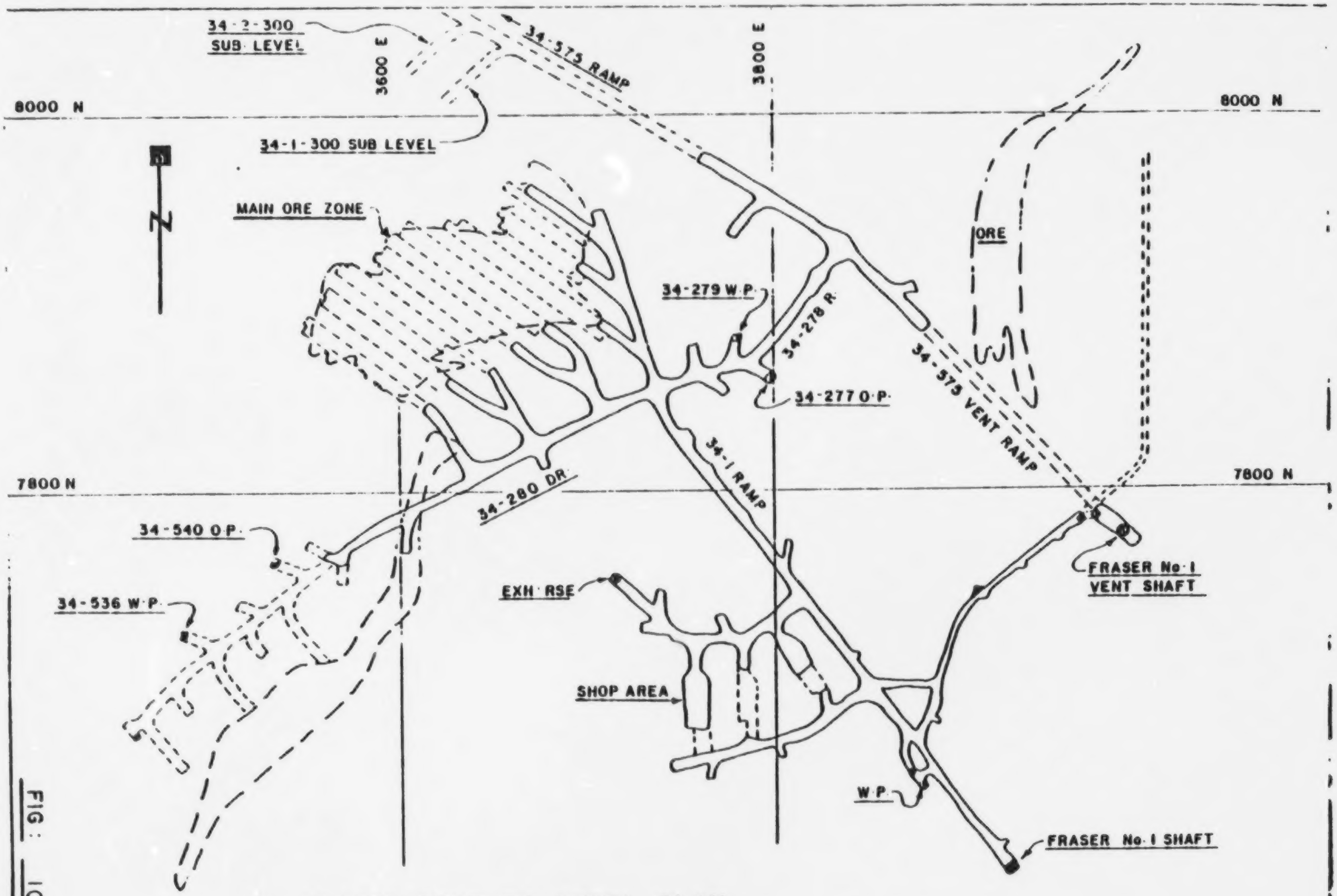

34-2-300
SUB LEVEL
34-575 RAMP
3600 E
3800 E
8000 N
8000 N
34-1-300 SUB LEVEL
N
MAIN ORE ZONE
ORE
34-279 W.P.
34-278 R
34-277 O.P.
34-575 VENT RAMP
7800 N
7800 N
34-280 DR.
34-1 RAMP
34-540 O.P.
34-536 W.P.
EXH. RSE
FRASER No. 1
VENT SHAFT
SHOP AREA
W.P.
FRASER No. 1 SHAFT

FIG: 10

FRASER MINE 34 LEVEL PLAN

FIG: 10

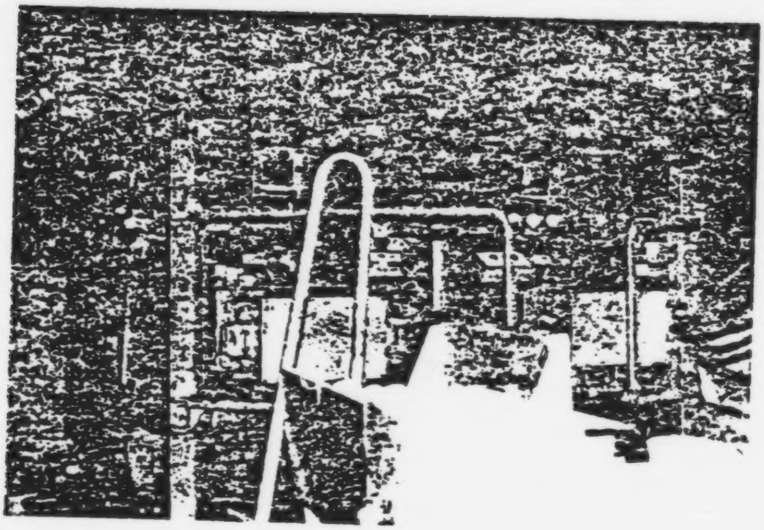


FIGURE 9 - ELECTRIC-HYDRAULIC DRILL JUMBO (Console)

TABLE 8

DEVELOPMENT ADVANCE RATES

	Hydraulic Jumbo	Pneumatic Jumbo
Meters of advance per available working hour	0.602	0.426
Monthly meters of advance	150	106
Yearly meters of advance	1,800	1,273

The maintenance costs in the original study were estimated to be 20% higher than for pneumatic equipment but our actual experience has shown that the maintenance costs at Fraser Mine are similar to those at East Mine; i.e. 50% less than pneumatic.

BITS AND STEEL FOR JUMBO DRILLING

The testing of different drill steel and bits presented some difficult problems. It was found that 25 mm hex drill rods were too weak and flexible. A few 28 mm hex drill rods were tested and it was found that these rods were suitable for smaller holes, (35 mm to 45 mm diameter), but these were not suitable for the hydraulic drills rated for larger diameter holes, (45 mm to 64 mm diameter). The testing also showed that 32 mm round rods were more flexible than 32 mm hex rods. The round rods tended to bend and snap more easily.

Rods with tandem thread (double thread length) at the bit end were tested in the hope that if the thread at the bit end was over-worn it could be cut-off and the unworn section of the thread could be used. In most cases this practice was not possible since by the time the rope thread was worn, one of the following would usually occur:

- the rod would snap from metal fatigue
- the full tandem thread was broken when attempting to remove a bit that was jammed to the rod
- the rod would jam in the face and could not be removed

Traditionally, rope type thread has been used underground and the following observations were made on its performance with hydraulic drilling equipment:

1) The bit end of the drilling rods tended to wear quickly due to its small diamter (32 mm) and as a result, bit removal was often very difficult and time consuming.

2) Couplings of a rope thread must be checked frequently for wear since over-worn couplings tend to bind to the rods and/or shanks. Often, the only way possible to remove the jammed coupling is to hit it with a sledge hammer. This results in damage to drill seals, shanks and drilling rods.

In view of these problems, an alternative type thread was investigated. It was found that a T-type thread, which has been successful on surface drill rigs, provided features that would eliminate problems encountered with the conventional rope thread. Although this thread was supplied by many manufacturers, the smallest size available was the T-38 version (this being a 38 mm diameter thread), and the smallest bit it could accommodate was a 51 mm diameter bit.

In order to obtain similar thread characteristics for 45 mm diameter bits, an independent local supplier designed a T-32 thread. When using this T-thread, bit removal was much easier and faster and at no time was a sledge hammer required for bit removal. The thread also showed very little wear after extended life. Very little interest was shown by the major bit manufacturers in investigating this type of thread, because they have a warehouse full of drill steel to sell with more conventional thread.

In the majority of our jumbo drilling, button bits of various makes have been used. At Fraser, 45 mm bits are routinely giving 70 meters/bit in abrasive granites. Better performances are being achieved at East and Strathcona Mines with identical make of bits. Up to 140 m/bit has been attained at East Mine. The one pass nature of button bits combined with its longer life, makes these bits more effective than conventional cross-bits.

PROBLEMS WITH THE INTRODUCTION OF HYDRAULIC DRILLING EQUIPMENT

With the introduction of hydraulic drilling equipment, it was discovered that attitudes and habits must change.

Improvements had to be made to the training courses, the mine planning, and the handling of spare parts. The mechanical workforce had to be increased. In multiple face development situations, the mining crew was strengthened by upgrading the leaders' rate, and increased supervision permitted better co-ordinated activities.

The lack of training for operators and mechanical personnel became evident very early by the low availability of the units and the time taken to get rounds drilled off. As a result of this, the various suppliers who had equipment on site were informed that their initial training programs were inadequate and that they must be prepared to offer extensive training services.

Mine operating personnel must be made receptive to change. Some older miners tended toward the attitude that hydraulic drilling was too complicated and that their methods of the past 25 years were good enough. A particular problem was encountered with the changing to button bits and consequent larger hole diameter (45 mm vs 32 mm for regular jackleg or longtom). Although the drillers were instructed to drill fewer holes, they insisted on drilling the same number. As a result of excessive holes, rounds were breaking poorly with considerable bootlegging. It took some time for the drillers to change ingrained habits; to accept that fewer holes were required; and that proper sequence for drilling the round entailed starting with the lifters.

Problems encountered with maintenance personnel were overcome by taking advantage of hydraulic courses and upgrading the training courses offered by suppliers. One manufacturer supplied service personnel on both shifts for several months to provide on-the-job training to our mechanics. This on-the-job training resulted in other mechanics taking on an interest in hydraulic equipment and as a result, maintenance in general was improved.

It is our conclusion that for electric hydraulic equipment the upgrading of maintenance personnel should be given the first priority. This training involves many manhours, but it pays off in subsequent high mechanical availability being maintained on hydraulic drilling equipment with minimal effort.

Another fact to be noted with regard to maintenance is that clean workshops must be provided to overhaul hydraulic drills; the fine tolerances of machine components mean that even minimal contamination may cause damage. Hydraulic drills cannot be overhauled on the muck pile!

Increasing the mechanical work force also aided in improved availability of the hydraulic drilling equipment. Although the hydraulic drill jumbo required an extra mechanic, it must be recognized that in most applications the mechanical work force would have been increased regardless because more mechanized equipment such as scoops, utility vehicles, etc., were also coming onto the site.

It was also quickly recognized that closer attention must be paid to scheduling in order to ensure high unit utilization. Production and maintenance planning must be co-ordinated to ensure that delay times are minimized. This means that the hydraulic alternative is less flexible and more applicable to multiple face headings.

Problems were encountered with the handling of spare parts. The method of handling spare parts through consignment stock stored on surface was found to be unsatisfactory because:

1) no surface warehouse was on site at some mines
2) some mines were not equipped for a warehousing system by which a constant record is maintained
3) inadequate storage space resulted in some parts being lost, and consequently, duplicate parts were ordered in some instances

As an outcome of this experience, it was decided that an underground stock be maintained at the mine and a good backup stock be placed in the central warehouse. As a result of this action, the parts problem was greatly alleviated.

In summary, the goal of maximum utilization of hydraulic drilling equipment requires genuine team work among all departments: mine operating, mine maintenance, purchasing, warehousing and mine engineering.

ENVIRONMENTAL FACTORS

Environmental improvements have resulted from the change to hydraulic drilling. The elimination of the oil laden exhaust fog has resulted in better visibility, cleanliness, and an improved atmosphere for breathing; the reduction in noise level has reduced operator fatigue.

Noise reduction is achieved in the low frequency bands due to elimination of the exhaust inherent with pneumatic drills. The noise in the high frequency area which results from the impact of the rock drill piston against the shank adapter remains. As a result of moving of peak sound levels from the 100-1000 Hz frequency to the 1000-8000 Hz frequency, an approximate 40% increase in the sound attenuating efficiency of most reputable personal ear protectors has been achieved. Therefore, not only is the hydraulic drill quieter, but the nature of the noise is easier to attenuate by personal protection. Hearing loss in an individual usually begins at 4000-6000 Hz due partly to outer and inner ear resonance amplification of frequencies at 3000-4000 Hz. Therefore, damage to the ear can still result with hydraulic drills and personal ear protection must be used (^{2}).

ENERGY EFFICIENCY

Another major advantage of hydraulic drilling is that it is more energy efficient. The piston design permits optimal transmission of energy to the rock face, hence less power is required to do the same work as pneumatic drills. From studies done by manufacturers and other mining companies, power costs for hydraulic drills are approximately one half of those of pneumatic drills. With only hydraulic drills in use, there is no longer a need to operate large surface compressors. Small portable compressors can be used for miscellaneous air needs such as charging ANFO into the hole. This will result in considerable savings in capital costs.

It should be noted that the greater the distance away from the compressor, the poorer the air quality and drill performance. Line losses can be minimized in transmitting electrical energy to hydraulic drills so that their performance is not affected. Table 9 gives a comparison in power consumption between hydraulic and pneumatic equipment for single drifters.

TABLE 9

POWER CONSUMPTION COMPARISON FOR SINGLE DRIFTERS

	HYDRAULIC	PNEUMATIC
Meters drilled per year	74,000	74,000
Average penetration	1.07 m/min	.46 m/min
Actual drilling time	1143 hrs	2667 hrs
Installed kw/drill	50	70
Consumption full load	57,150 kwh	186,690 kwh
Consumption no load* (20% of full load)	11,430 kwh	74,676 kwh
Total energy consumed/year	68,580 kwh	261,366 kwh
Power cost at $0.05/kwh	$3,429	$13,068

*The no load operating time of compressors is high for production equipment and is taken at twice the actual drilling time. With the hydraulic alternative, the rig operates on no load only during rod pulling, collaring and setting up time.

SUMMARY AND CONCLUSIONS

Falconbridge's experience, as discussed in this paper, has clearly demonstrated that hydraulic drilling is viable in terms of economics, productivity and safety and health considerations. It has also been shown that the hydraulic drills can be employed in a variety of situations in development, stoping, longhole drilling and large hole drilling for blasting. The following summary of performance data shows that these machines are far superior to the pneumatic drills.

	Hydraulic	Pneumatic
Development Jumbo		
Average drilling rate m/min	0.82	0.21
Average advance rate m/month in 3 to 4 headings	133	88
Time to drill off a round; % of shift	50	100
Maintenance costs/m drilled	1/2 of pneumatic	
Longhole Drilling		
Average meters drilled/m.s.	70	25
Maintenace costs/m drilled	90% of pneumatic	
Large Hole Diameter Drilling		
Average meters drilled/m.s.	46	30
Maintenance costs/m drilled	same for both	

Table 10 summarizes hydraulic drill results at East and Fraser Mines compared to pneumatic drill results at Strathcona Mine.

TABLE 10

JUMBO ADVANCE RATE COMPARISON

(5 m x 4 m DRIFT)

	2 Boom Pneumatic Jumbo	2 Boom Hydraulic Jumbo	
No. of Headings	Avg m of Advance/month (Strath. Mine)	Avg m of Advance/month (East Mine)	Avg m of Advance/month (Fraser Mine)
1	45	-	-
2	59	88	87
3 to 4	88	116	133

Savings in energy costs are significant. This could further affect the development of a new mine where capital costs in a compressor could be reduced.

In order to exploit the maximum potential of the electric hydraulic machines, it is of vital importance that attention be paid to the following:

1. Training of the operators and maintenance personnel

2. Technical support services to be provided by the suppliers

3. Resistance to change is a major psychological barrier

4. Traditional methods of mining have to change to accommodate hydraulic equipment

5. Planning and scheduling must be coordinated in order to provide high utilization

6. All departments must work as a team

FUTURE PLANS

As a result of our experience to date a firm commitment has been made to stay with hydraulic equipment. Two development jumbos have also been introduced in our Onaping Mine. One of these, a 2-boom rail bound jumbo, is part of all electric high speed drift drive in a single heading. The other jumbo has been acquired for trackless development.

We are in the process of evaluating small hydraulic machines to replace jacklegs and stopers. We are also seriously considering the introduction of a fully automated electric hydraulic roofbolter.

ACKNOWLEDGEMENT

The authors would like to thank Falconbridge Nickel Mines Limited for permission to publish this paper. This task was accomplished as a team effort by various departments. Special acknowledgement is given for the contributions made by the mining team, and various colleagues who provided valuable criticism in preparation of this paper.

Acknowledgement is also given to the various equipment suppliers for their assistance.

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APPENDIX 1

1. Present Value of the Costs

The present value of the cost (operating and capital) was computed by discounting the cash flows at the internal cost of capital.

For cash flows (i.e. after taxes, annual operating and capital costs) calculations, the following criteria were used:

- Five years useful life
- No salvage value
- Straight line depreciation
- Fifty percent taxation rate

2. Net Present Value

The net present value is the net difference between the present values of the annual cash outflows and inflows.

3. Internal Rate of Return (I.R.R.)

The internal rate of return is that rate which equates the present values of the annual cash outflows and inflows.

4. Payback Period

The payback period is equal to the number of years necessary to recover the initial cash outlay.

Computer control com

Kristen Dahl, Vice President, Ingeniør Thor Furuholmen A/S, Oslo

Electronics and computer technology can be expected to influence productivity and economy in the field of construction equipment in the same way that manufacturing and process industries have already been affected.

Looking back a few years, the first really significant change to affect drilling jumbos was the introduction of the hydraulic percussion hammer in the early 1970s. This started to replace the old pneumatic type, producing great improvements in environmental conditions at the face: less noise, less dust and the absence of the unpleasant oil/air mist associated with pneumatic drilling. But even more important, the hydraulic drills had significantly higher penetration rates, up to twice as great as their pneumatic counterparts, and consumed less energy per metre drilled (see *Tunnels & Tunnelling*, December 1979, p5 and p14).

Hydraulic hammers can save money

As always, doubts were raised about reliability—hydraulic systems are prone to fail if dust gets into the system, and special precautions had to be taken, including the use of dust-free workshops for maintenance—and about whether the equipment was robust and sturdy enough for the rough handling of perhaps inexperienced labour on remote overseas sites. The investment required for hydraulic drills is also much higher and at first their total economy was questioned until it was proved by experience that they really could save money. Now there is little doubt that hydraulic hammers will eventually take over the entire market.

Although the developments mentioned have brought the drilling process to a high level of mechanisation, there are still a number of shortcomings. In practical tunnelling situations the following points may be observed:

First, although a detailed drilling and blasting plan may as a rule have been worked out, the operators make their own drilling pattern, which they all believe is the best one, although they may be quite different in position and number of holes from gang to gang, and all different from the official one.

Secondly, contour holes have often been placed at random, whereby they, together with overloading, have increased the overbreak unnecessarily. The direct cost of overbreak is limited, but its effect on stability can result in high extra costs. If the tunnel has to be concrete lined, this becomes evident, as the total overbreak volume has to be replaced with concrete.

> Electronics and computer technology are rapidly spreading into the field of tunnel construction. Microprocessor controlled hydraulic drilling jumbos have been successfully used by a Norwegian contractor to drill in excess of half a million metres of hole, and are now ready for series production.

Thirdly, on multiboom jumbos, one of the booms is often idle while the operator attends to some other problem. Also, the time spent moving from one hole to the next depends very much on the operator's skill. This greatly influences the total drilling time.

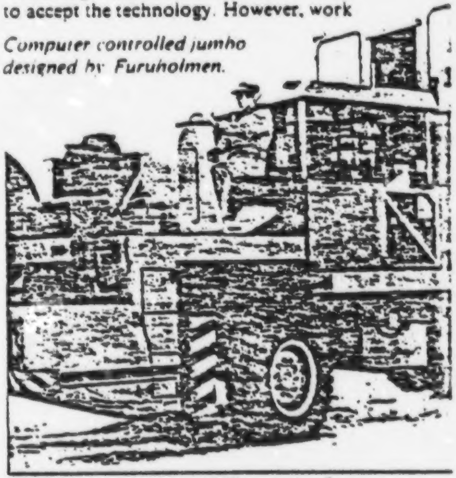
Fourthly, the penetration rate of the drill hammers depends very much on the adjustment of parameters like percussive energy, feed pressure rotation speed, etc. Optimum working conditions vary along the tunnel, from hole to hole, and sometimes even within the length of each hole. Therefore the hammer parameters should actually be under constant surveillance and modification, which has been impossible to attain up to now. From the highly variable results observed on the same hammers under assumed equal conditions it can be deduced that they frequently work far from their optimum adjustment.

Development programme for automation

Finally, rather rough treatment of the equipment, for instance very abrupt boom movements, is a common sight in tunnels.

The Norwegian engineering and contracting company Ingeniør Thor Furuholmen A/S had forseen as far back as 1970 that electronics and computer technology would one day affect its main business of hardrock tunnelling and so in 1972 started a development programme for automating tunnel drilling rigs.

During the study, an ordinary full scale one-boom rig was mounted in the laboratory equipped with instruments and a computer and used for test runs of mock positioning and drilling. It was concluded that, from a technical point of view, computer controlled tunnel drilling rigs could be developed with the specified performance and reliability, but the equipment cost was high and at the time it was thought that the market was not ready to accept the technology. However, work was started again in 1977 and the first working prototype, a three-boom jumbo, was produced and tested by 1978.


Computer controlled jumbo designed by Furuholmen.

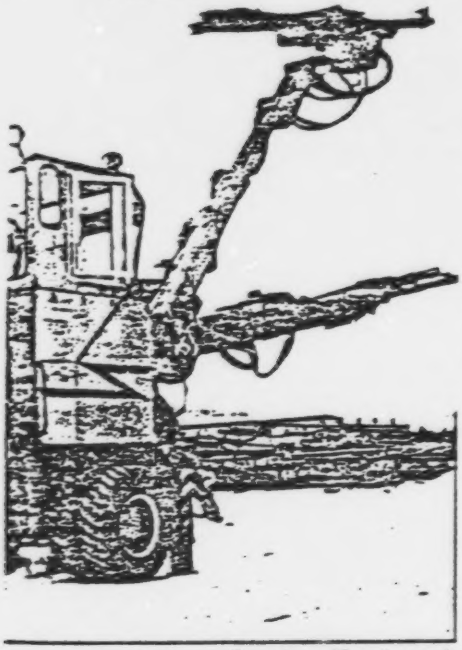
At the present time the company's experience includes drilling more than half a million metres of 45mm holes using the micro-processor controlled prototype drilling jumbos. The latest versions are equipped and programmed for automatic and precise drilling of complete rounds according to predetermined drilling plans defined by fixed coordinates. The microprocessor-based control systems are now being licensed to manufacturers of equipment.

The computer and general electronics technology applied in Furuholmen's automatic tunnel jumbos have been developed by The Central Institute for

es to hard rock drilling

Industrial Research, Oslo, Norway, which has accumulated much experience from other industries such as ship building, paper processing and instrumentation of North Sea oil platforms.

Furuholmen started tunnelling in 1945 and since then has completed about 600km of tunnel at the rate of up to 50km per year as well as about 60 major rock caverns for hydroelectric power, oil storage, sewage treatment and other civil and military purposes.



At present the company owns a total of about 50 tunnelling jumbos of all types, most of which are of its own design and manufacture, and in addition three full-face machines owned and operated jointly with Prader of Switzerland.

Prototype development

When the project was started again in 1977 the market outlook was still unclear, but several conditions had changed in favour of this technology. The competition for skilled operators had increased. The cost of labour had increased, more than the cost of equipment. New government rules for safety and working conditions for operators, such as sound insulated operator cabins with clean air, had been introduced. Huge improvements in micro-electronic technology that facilitated cheaper and more reliable design of computer control systems for this purpose had by this time become available. Experience from the use of micro-processors had already been obtained from other applications.

It was concluded that these factors, together with Furuholmen's own need for new, modern tunnel drilling rigs, justified further development leading to working prototypes.

Design of the first prototype

For the first working prototype the aims were limited to having the physical equipment necessary for automatic control installed. The jumbo had however to be operated manually and run with only the measurement and display part of the programs active to test out reliability and suitability of the different components.

The main specifications were set as follows: the first prototype was designed with three booms to drill on face areas from 35 to 90m^2 in one set-up. The boom reach was 8.5m high by 12.5m wide. The boom geometry and chassis together with the hydraulic system were all of new design. Total weight of the jumbo was 42 tons.

In designing the booms, much emphasis was placed on getting a simple but rigid construction to facilitate ruggedness and precision. It is one of the benefits of using advanced control systems that the mechanical part of the system can be optimised for rigidness and mechanical simplicity without being limited to geometries that are easy to operate manually.

The booms each have seven independant axes of movements in addition to the motion of the drilling hammer on the drill guide. The sensors, measuring the motion of each axis with great precision, were designed as an integral part of the boom so that they are well protected against damage and humidity.

The hydraulic powerpacks were designed as modules, one for each boom with the same motor and pump unit for powering the hammer work as well as boom movements and with value manifolds comprising all the electrohydraulic control valves as an integral part.

The electronic operator panel, including a CRT graphical display and electronics cabinet containing the micro-processors and interface circuits, were installed unprotected on the rig, without any operator's cabin or other shelter.

The prototype was first used on a large sewage treatment plant at Slemmestad in the Oslo area, where a system of 12 large caverns for process equipment was drilled and blasted. With so many drilling faces available in a concentrated area, the rig was operated almost continuously on a two shift basis. During this period of about a year the rig drilled 380 000m of holes for blasting out approximately 270 000m^3 of solid rock (limestone and shale). This demonstrated a high mean availability of the rig. Maintenance problems in general were very small, for the electronic equipment and sensors as well as for hydraulic and mechanical parts.

When this job was finished, a thorough inspection and checkout of the whole rig was performed. It turned out that the wear and tear on the booms also was quite negligable compared with the general experience from other rigs. This is probably due to a great extent to the soft and controlled manoeuvering of the booms obtained through the electronic control system.

First prototype at work

The first prototype is now drilling on the mainland side face of a road tunnel under an arm of the Barents Sea out to the city and island of Vardø. Until now it has drilled roughly 100 000m in very fractured sandstone.

Extremely difficult geological conditions have caused great problems for the drilling and blasting. Concreting has been necessary to a great extent, and it has therefore not been possible to utilise the rigs' capacity in the same manner as on the first job. However, the cold and humid climate in this area, with a seawater infiltration through the tunnel roof, represents a severe environmental test of reliability for the electronics. So far all

Computer control co

parts of the jumbo, including the electronic part, have withstood the test.

Design of the second prototype

The second prototype was ready for testing in September 1979. Mechanically, this rig is identical to the first one with the same size of chassis, booms, hydraulic power units, etc. but several additional features were included. These were a sound insulated and heated operator's cabin, and the computer was programmed for automatic positioning and drilling of complete rounds in curved tunnels, as described later. After a testing and debugging period of two months, the rig was set in production on its first job.

The first job for this second prototype was two road tunnels in the Arendal/Grimstad area in southern Norway. Approximately 120 000m of hole was drilled in hard basalt stone. The cross-sectional area was 70m², and both of the tunnels were curved in the horizontal as well as the vertical plane, with variable cross sections.

As this job was a combined training and test-out of computer control under almost full production, the system was first operated in the manual computer controlled mode. Gradually automatic controlled positioning and drilling, according to the programmed drilling plan, was applied.

Satisfactory operation

Technically the equipment operated satisfactorily with few maintenance problems. After some adjustment of the procedures for automatic collaring and bending compensation, the automatic control mode was operated regularly with satisfactory results. The procedures for drilling curved tunnels according to the programmed plan worked perfectly with few problems in drilling the contour holes when successive rounds were drilled by program control.

When a round for some reason had been drilled manually, minor geometrical transition problems had a tendency to occur. Systematic statistics have not been collected so far, but there is no doubt that the contour was generally smoother and more regular when programmed drilling was applied.

After the job was finished, the whole system was carefully checked and the same conclusion could be drawn as with the first rig concerning wear and tear and maintenance cost. The next job for this jumbo is expected to be a hydro tunnel in Greece now under negotiation and planning.

A third prototype with the same equipment and specifications as number two was ready in January 1981, but has not yet been assigned to any job.

Two prototypes of small size jumbos have been built and are now on their first job on hydrotunnels at the Aurland power plant in the western part of Norway. These jumbos both have two booms designed for face areas from 10 to 20m². The booms have in principle the same type of geometry as the three big prototypes, but dimensions and weights have been reduced. The booms have been designed and prepared for computer control. They are, however, at this first stage operated manually from an electronic control panel.

Description of equipment and methods

The control system consists of:

- Electronic control unit with printed circuit boards based on microprocessor technology
- Operator's panel with easy-to-learn joy-stick controllers
- Visual display unit for graphic display of drilling plan; drilling directions relative to selected axis; drilling penetration rates; alarms and other operater and maintenance aids.
- Sensor units for measuring positions, angles, pressures, etc
- Hydraulic power pack with electrohydraulic control valves for controlling boom movements and drilling.

The control system is based on a modular design and can therefore easily be adapted to rigs with a different number of booms, different geometry of the booms, different drillhammers, etc. The electronics, cabinets and sensors have been specially designed to survive the rough environmental conditions found in tunnels.

Method of operation

The jumbo can be operated in three different modes:

(1) Automatic positioning and drilling according to a pre-programmed drilling plan
(2) Manual operation by using the manipulator mode of the control panel
(3) Direct control, which is a standby option to enable it to be operated even if a failure should occur in the control system.

The mode of operation can be selected individually for the different booms by means of push-buttons on the operator's panel. The design of the control system makes it easy to choose between the different modes and to change from one to the other.

Manual override in the automatic mode can therefore easily be performed, for instance in case a hole cannot be drilled in the planned position. After the manual override is performed, the system is set back to the automatic mode again by a push button

Automatic positioning and drilling

In the automatic mode the control system moves the boom to the programmed position for drilling a hole according to a programmed drilling plan. Then the feed rail is moved towards the rock and drilling is started, entering the drillbit smoothly, so that no side-sliding occurs. After the drilling is controlled to programmed depth, the drill is returned and the boom is automatically moved to the next hole position and the procedure is repeated.

The programmed drilling plan is displayed on the Visual Display Unit (VDU) so that the operator can follow the progress of the drilling at all times. Separate symbols are used to mark the holes that have been drilled, the hole being drilled and the next hole to be drilled, so that the operator can override if any obstacles should make this necessary.

If, for some reason, the hole cannot be drilled in the planned position, the operator may override and move the drillbit to a better position by means of the manipulator position joy-stick. The direction of the drillrod will automatically be parallel to the programmed one.

If the operator presses the button ORIENTATION, the drillrod will change direction so that although it points toward the bottom of the hole, which will remain as planned, the start of the hole will have been manually moved to a new position.

[illegible]

Precise drilling without precise positioning of the rig or marking on the tunnel face may be carried out as follows: the different blast holes are programmed in a local coordinate system in relation to the tunnel axis at the face. The description of all the holes relative to this coordinate system constitutes the 'drilling plan'. All the holes can have individual positions, directions and depths.

The local coordinate system is described relative to the fixed coordinate system as a function of tunnel length by the 'tunnel description'. The fixed coordinate system is defined by the direction of a laser beam and the direction of gravity. The 'tunnel description' and the 'drilling plan' are stored in the control system's memory.

Several alternative drilling plans can be stored simultaneously and selected by a button. New 'drilling plans' and 'tunnel descriptons' can be applied by means of a magnetic tape cassette in the field.

Position of blast holes computed

After the jumbo has been moved to its working position in front of the face, the position of its local coordinate system relative to the fixed system is measured. This is performed by moving one of the drill guides by means of the manipulator joy-sticks until it coincides with the reference laser beam. When the 'navigation' button is pressed the computer will then read the different boom-angles together with the rig's tilt-angle and compute the rig position. This information is stored and used to compute the position of the blast holes relative to the rig's coordinate system.

The necessary angles and boom-extensions to position the drillrods for drilling of a certain hole in the drilling plan are computed by means of a mathematical model of the geometry of the rig and the booms. This mathematical model is operated dynamically in real time, so that the motion of the boom from where it actually is to the programmed drilling position is smooth and well controlled.

Manipulator mode

Precise positioning and directioning of the drilling tool is a difficult task on an ordinary jumbo, because each individual motion influences the position as well as the direction of the rod. On a rota-boom this is especially complicated. In the MANIPULATOR mode precise manual control of the drill guide is a very easy operation and the boom movements are taken care of automatically through the computer. The control is performed by means of two three-axis joy-sticks. One joy-stick is for change of position and the other for change of direction of the drillrod.

The coordiantes of the joy-sticks are relative to the positon and direction of the drillrod, so that the operator does not need to think of the complicated geometry of the boom.

The direction of the drillrod is shown graphically on the VDU relative to the predetermined reference axis. Initially the reference axis is parallel to the axis of the rig. By setting one of the drillrods in a new direction and pressing the navigation button the direction of this drillrod will be the new reference axis.

When pressing the orientation button on one of the other booms, this boom will automatically change its direction to be parallel to the reference axis, without changing the drillbit position.

The manipulator mode is convenient to use when overriding manually in the automatic mode, positioning one of the feeds parallel to the reference laser beam for establishing the rig's position; or for every drilling task where a preprogrammed plan is not used.

Advantages

Based on experience so far, Furuholmen is confident that the equipment and methods work reliably in the rough environmental conditions encountered in tunnelling. Improvements in operational performance and drilling precision have been demonstrated. Maintenance of the booms and associated parts has been greatly reduced due to the smoother action of the computer controlled movements.

The computer controlled drilling rig is also a tool with the inherent possibility of working systematically towards optimum results for hole patterns, hammer work, etc. This comes from the fact that the whole work procedure is recorded by the computer so that results can be measured against choice of parameters. This means that the tunnel engineer takes a well equipped laboratory with him into the field.

Future developments

Although the systems described are technically advanced, they represent only the beginning of the application of sophisticated techniques in tunnelling. It is an inherent feature in the design of the systems that further developments may be added later.

Further developments envisaged in drilling include automatic drilling without supervision, optimisation of hammer-work, and automatic monitoring and indication of equipment failure.

Among other applications in tunnelling, scaling and shotcreting could be mentioned as interesting areas.

The methods and equipment developed are now being prepared for series production, and will be marketed world wide. Licence agreements with rig manufacturers are being negotiated for application to their products for mining and construction. Furuholmen, as a contractor, will offer demonstrations of equipment and methods during the performance of actual tunnelling jobs. ☐

For more information enter 101 on reader enquiry card.

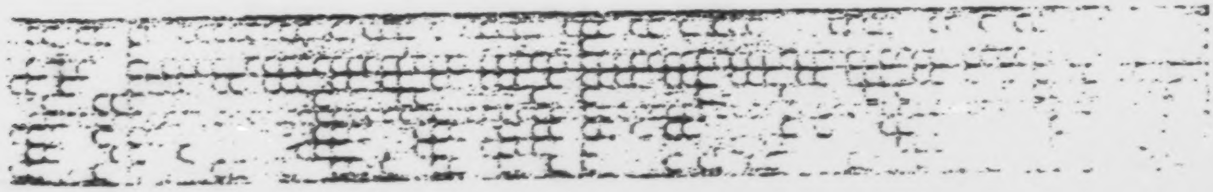
The development and manufacturing of the equipment and methods for computer controlled tunnel drilling are performed by the following group of companies and research institutes:

Ingeniør Thor Furuholmen A/S—
A Norwegian civil engineering contracting company. In tunnelling work they are one of the biggest operators in northern Europe with approximately 40 jumbos and three full-face machines in operation.

Andersens Mekaniske Verksted A/S—
A Norwegian machine shop specialising in design and manufacture of equipment for mining and tunnelling.

Central Institute of Industrial Research
A Norwegian industrial research institute, which has a wide experience in the development of industrial automation systems for any purpose (including rough environments), based on advanced technology.

Norwegian Electronics A/S
A company specialising in development and manufacture of professional electronics for industrial purposes.



MINING METHODS

Down-the-Hole Drilling at Geco

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Manitouwadge, Ontario

Presented at the CIM Underground Operators Conference,
Sudbury, Ontario, February 1981

The present and future success of the Geco operation depends to a great extent on the ability to drill, load and blast the ore remaining in pillars. Due to ground movement caused by the mining of the primary 70-foot transverse panels, the remaining 120-foot pillars are generally cracked. Continued movement during the blasting of the required void in the pillar will not allow the successful loading of traditional blastholes. 4.5-inch-diameter blastholes, up to 130 feet in length, were successfully used in these areas. To date, approximately 330,000 feet of 4.5-inch hole has been drilled, of which 288,000 feet have been blasted to produce 1,342,000 tons of ore.

Location

The Geco operation of Noranda Mines Limited is located in Manitouwadge, Ontario, at the end of Highway 614, some 54 kilometres north of its junction with the Trans-Canada Highway, No. 17. The junction with Highway 17 is located mid-way between Sault Ste. Marie and Thunder Bay.

Geology

The Geco orebodies are located in a Z-shaped dragfold on the south limb of the Manitouwadge Syncline. All known orebodies on the property are in a highly folded quartz-muscovite schist zone. The Geco main orebody plunges 35 degrees to the east, parallel to the dragfold, with a near-vertical dip. Local pinch-and-swell features cause local changes in strike, dip and thickness of the ore zones. The main orebody has a strike length of approximately 2500 feet and varies in thickness from 150 to 200 feet in the west end, near the drag fold, to 10 feet or less at the east end.

The main zone is composed of a massive sulphide core containing pyrite, pyrrhotite, sphalerite, chalcopyrite, galena, silver and 10-15% siliceous gangue enclosed in a disseminated zone of pyrite, pyrrhotite, chalcopyrite and 60-70 per cent siliceous gangue, mainly quartz-muscovite schist.

The orebodies have been locally faulted and cut by diabase dykes.

Mining

Since the beginning of operations in 1957 until the end of December 1980, the operation had produced 34,530,000 tons of ore grading 1.91% Cu, 3.34% Zn and 1.74 ounces of silver per ton. Ore reserves as of December 31, 1980, were 20,870,000 tons grading 1.85% Cu, 3.72% Zn and 1.46 ounces of silver per ton. Approximately 14 million tons of the remaining ore reserves are located in the transverse pillars. By the end of 1980, all transverse primary stopes had been mined.

Approximately 90 per cent of the production has come from sub-level blasthole stopes, with the remainder coming from development and cut-and-fill stoping.

Because of the geometry of the orebody, it has been divided into two types of blasthole stopes (Fig. 1). The western or lower portion of the orebody is divided into transverse 70-foot primary panels separated by 120-foot transverse pillars, with 150-foot transverse boundary pillars at desired locations. The upper or eastern portion of the orebody has been mined by both longitudinal blasthole and longitudinal cut-and-fill stopes. Cut-and-fill stopes are only utilized where ground conditions will not allow some form of blasthole stope to be used.

A primary transverse blasthole stope was generally mined in two 400-foot-high lifts, with the top lift removed first. The ore was developed by perimeter-type sublevels spaced at 100-foot intervals in height. The sublevels were connected by one or more slot raises. After approximately 25 per cent of the ore had been removed to create a void, the remainder of the stope was blasted in one shot. As soon as the final shot was blasted, waste rock from the surface quarry was introduced into the stope. This addition of waste rock on top of the broken ore was necessary to control ground movement both in the waste and pillar walls. As shown in Figure 1, the No. 4 waste rock fill system serves those stopes in the eastern portion of the mine.

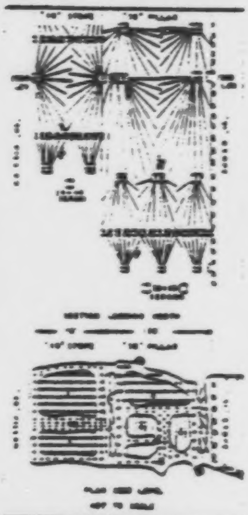

FIGURE 2. Plan of the 2280 level

Other fill systems serve the western portion of the mine. The ore-waste interface was maintained horizontal by means of a draw control program. The bottom lifts were mined in the same manner as the top lift. When the panel was completely mucked out, bulkheads and drainage outlets were installed in all openings and a mixture of mill tailings and cement, at a ratio of 30:1, was introduced into the stope. This consolidated the waste rock, thereby allowing almost 100 per cent recovery of the adjacent pillars. Only where the cemented tailings had not percolated down along the pillar wall, thereby leaving areas of unconsolidated waste rock, has any significant run of fill been noted in the blasted pillars. The pillars are mined by the same method as the primary stopes, except that they are generally mined in three lifts. This is done for better draw control on the broken ore between the two fill walls.

Longitudinal blasthole stopes are mined out and then filled with either straight mill tailings or cemented tailings. If any sloughing does occur in these longitudinal stopes, a draw control program is used to remove the ore from under the slough. The slough remains in the stope as fill.

History of Blasthole Drilling

From the start-up in 1957 until March of 1975, all blasthole drilling was performed with conventional percussion drills utilizing a 4-foot rod change and 2½-inch bits. Initially, all machines were on a bar-and-arm setup. In later years, many types of drill carriages were tried, with the result that Atlas Copco developed a "Big Hole Wagon" to carry the larger of the units Geco was operating. Except in special cases, all longhole drills are operated by one man.

The 49 panel, as shown in Figure 2, is a primary panel and is typical of the 2½-inch hole patterns. The standard ring burden was 8 feet, with a toe spacing of approximately 10 feet. The hole lengths were kept below 70 feet by means of fan drilling. The up and down holes were overlapped 5 feet to ensure complete coverage. The 12-foot-wide sills were drilled on a 4- by 4-ft square pattern. All flat rings had an 8-foot ring burden and a 12- to 15-foot toe spacing. All undercut rings had a 4-foot ring burden and a 6- to 7-foot toe spacing. The 2½-inch holes were very successful in the primary panels where no ground movement was encountered.

In order to ensure an orderly and efficient mining sequence for the entire orebody, pillar recovery was started in 1972. It was then found that the small holes, which had worked so well in the past, were not as effective in the broken and stressed ground encountered in the pillars.

Pillar Recovery

The mining of the primary stopes caused two significant problems in the pillars. (1) The blasting in the primary panels caused a fracture zone to develop in the east and west edges of the pillars. This zone may extend up to 10 or 15 feet into the pillar. (2) The removal of the primary panel caused a relaxation of the waste walls and thereby developed a zone of shearing along north and south limits of the pillars. It is important to note that all the Geco pillars have movement within their boundaries from the time the void blasting commences until the final blast is fired. The highly schistose nature of the wall rocks and disseminated ore zones localize this movement to the area between the massive sulphides and the north and south limits of the pillar.

The development of the pillar, which includes sublevels and raises, coupled with the effects from the void blasting, magnified the above problems. All holes drilled prior to the completion of the void blasting are subject to closure either through squeezing or offsetting. Those along the perimeter of the pillar are most prone to this problem.

In order to clean the affected 2½-inch holes, extensive blowing, washing, rodding and/or redrilling was required to allow loading. Many stopes have had their on-stream dates deferred by as much as three months to allow for hole rehabilitation. In many instances, holes were reamed to 3 inches in an effort to keep them open for the time required.

To overcome the hole closure problem, it was contemplated to do only that drilling required for the void, blast the void and then complete the drilling of the remainder of the pillar. This exercise was found to be too time consuming and would have set the production date back farther than the time required for redrilling. During the search for a way to overcome the hole closure problem, it was noted that the upholes, if they were not offset, were much easier to clean than the downholes because of the effect of gravity. It was also noted that the 3-inch holes remained opened longer and were easier to clean than the 2½-inch holes.

In early 1975, the belief was that if it were possible to drill all larger-diameter downholes with the majority of them being drilled to breakthrough, the cleaning problem would be greatly decreased.

Down-the-Hole Drilling

In March of 1975, Geco purchased its first down-the-hole rig. This was an Atlas Copco COP 4 machine utilizing 4½-inch button bits. Results from this machine indicated that 4½-inch holes could be drilled, cleaned and loaded in areas which could not be extracted using 2½-inch holes. This large hole size is also economical in other areas when the loading and blasting portion of the operation is considered. (See Table 2.) It was also found that deviation of the holes was less than 2 per cent, whereas the deviation in the 2½-inch holes could be as high as 5 per cent. The 4½-inch holes which were drilled to breakthrough were easy to clean and usually did not require any cleaning. Those holes not breaking through could be cleaned by one or more of the standard methods. Some holes have to be machine cleaned. Machine cleaning may be done with or without a hammer and bit on the drill string, depending on conditions. At the time of writing, research work was being done on utilizing a 3½-inch rotary bit for hole cleaning where machine cleaning is required.

It was also found that when drilling there was less sticking of rods and bits with the down-the-hole hammer. During normal drilling operations, a small amount of water is injected into the system for dust control. However, when a hammer or bit gets too badly stuck, the drill string, hammer and bit are lost in the hole. In badly fractured ground, the difficulties of stuck bits and hammers has been partially overcome by means of drilling dry with the aid of a dust collector or by introducing a foaming agent in the flushing water. At present, only foam is used because a satisfactory dust collector which will fit into our headings has not been found. Geco utilizes an [illegible] piston pump for the injection of flushing fluids into the system. Because bit life and penetration is 20 per cent greater when drilling dry, research is still underway to develop a satisfactory dust collector. The down-the-hole hammer is less noisy than the conventional hammer and driller acceptance is very good.

The average bit life per stoping area varies from 450 feet in the east end of the mine, where the ore is locally of the disseminated type, to 900 feet in the west end of the mine, where there is a high proportion of massive sulphides. The mine-average bit life is 750 feet. The greatest problem with bits is gauge loss caused by ground conditions and over-rotation. It has been found that at low rotation speeds (10-12 rpm), the air-powered rotation unit gives a very jerky rotation. When the bit is operating properly, good-size fragments are broken from the face of the hole and these fragments tend to jam between the sides of the flushing groove and the hole wall. This problem is compounded in broken ground. Therefore, to void problems of this nature, over-rotation is the standard practice. All bits are sharpened in the working place by the driller. Penetration varies from 16 to 25 feet per hour, depending on location and ground conditions. The penetration rate is based on drilling time only and does not include rod additions, etc. See Table 1 for details.

Deviation in the hole itself is not a problem, as most holes less than 65 feet in length are straight enough to hole through. In most cases where holes do not break through at the desired location, the problem can be traced back to setup errors. When drilling parallel holes, such as in drop raises, a bar-and-arm setup is preferred.

The plugging of the holes as soon as they are drilled, with wood or plastic plugs, is critical to keep the holes from becoming filled with sludge. In areas of broken ground, fractures which connect the hole being drilled to those already drilled cause a real problem which has not yet been overcome. Ring drilling is very susceptible to this problem.

Moving between rings utilizing the Big Hole Wagon can generally be completed in 15 minutes or less providing the distance between the floor and back is greater than 8.5 feet at that location. Bar-and-arm setups take slightly longer.

Maintenance on the units is very low, therefore giving good availability. In general, this unit will accept a fair amount of driller abuse as long as dirt is kept out of the lubricant, air, water, rods and hammer. (See Table 1.)

High-Pressure Drilling (230 psi)

In 1980, a booster compressor was purchased to boost the mine air to 230 psi at a rate of 420 cfm. This compressor operates one COP 4 rig. This unit has increased drilling rates proportional to the air pressure increase. Bit life is approximately 20 per cent better than with the average mine air. Because the average penetration rate in the massive ore is 75 feet per hour, a one-man operation is not feasible over extended periods of time. Therefore, this and the next two high-pressure units purchased will be used in the east end of the mine, where the average penetration rate is 35 to 45 feet per hour for high-pressure drilling. (See Table 1.)

Down-the-Hole Drilling Patterns

Figure 2 is a combination of the blasthole drilling layouts used for extraction of a portion of the 49 primary panel and the 50 pillar. As mentioned previously, the 49 panel was extracted by 2½-inch holes.

The 50 pillar utilized the 4½-inch hole with a maximum 10-foot ring burden and 12-foot toe spacing. All holes are drilled below the horizontal. No holes are drilled closer than 4 feet to the fill wall.

The standard slot is drilled 10 feet wide, with three parallel rows of holes, as shown in Figure 3B. The majority of the slots require extra holes around the slot raises.

We have experimented with a herringbone pattern, as shown in Figure 3C, for drilling blastholes in the 44 pillar. The pillar was divided into two panels, each 50 feet wide. This pillar width did not allow more than one 9-by-9 subdrift to be driven as a drilling base. Where the slot raises are not vertical, extra holes were drilled in the slot to bring it to vertical because all rings are vertical. This pattern gave good results.

Narrow Ore

A standard pattern, as illustrated in Figure 3A, has been developed and used in the narrow ore zones. Regardless of the ore width, a trench with a minimum width of 6 feet is drilled and blasted. The rings are vertical, and the tip of the holes is adjusted to cover the ore outline at breakthrough if it is wider than 6 feet or if the ore is not standing vertical. The 4½-inch holes are hard on the walls if drilled on the ore/waste contact. However, if the hole lengths are over 45 feet we use the large hole because of accuracy and the extra powder we can put into the hole to ensure that the trench stays [illegible].

Drain Holes

Geco has had very good results using the COP 4 for drilling drain holes up to 220 feet in length. Accuracy is within 4 per cent, with only two holes being outside this. Of the 26 holes already drilled, only one has been lost. This was because of a combination of setup error and a change of ground hardness over the hole length. Best results are obtained if the holes are near vertical and the ground is very hard.

At present, we are researching the feasibility of using one or more 4½-inch holes to ventilate our Alimak raises, which are 200 to 220 feet in length. The hole is used as a pilot hole as well as a ventilation hole for the raise. To date, only one raise has been driven this way, but results are encouraging. A 5-inch air mover on top of the hole is used to move the air.

Results of 4.5-inch Downholes

The 4½-inch downhole has had a significant impact on all forms of blasthole mining.

Transverse Pillars

A significant proportion of these pillars are now being blasted sublevel by sublevel, rather than with the original shell-type blast required when using the 2½-inch holes. The practice now is to completely drill enough sublevels in a stope to ensure that blasting will not have to wait for drilling. The usual blasting sequence coupled with continuous removal of broken ore to provide the required void for the next blast is as follows: blast the undercut, blast the slot raise to the next sublevel above, blast the required void for this interval, blast the remainder of the sublevel, blast the slot raise to the next sublevel and continue as before. At all times, the broken ore is left as close to the unblasted back as possible to minimize movement of the fill walls. It has been found that because the ore is left as long as possible in large undisturbed blocks, ground movement is minimized, thereby minimizing loading problems.

Longitudinal Blasthole Stopes

Originally, these stopes were mined by means of "vertical slices" which extended from the bottom to the top of the stope. These "slices" were retreated from one end of the stope to the other. Because the bottom portion of the slice had to be clear for blasting, there was very little broken muck left in the stope to support the walls. This allowed sloughing in the "[illegible]" and became a significant problem once the stope was opened up for a hundred feet along strike for the full height.

Using the waste-rock backfill theory of controlling wall sloughing, it was decided to blast one of these stopes sublevel by sublevel and leave as much broken ore in the stope as possible to stabilize the walls. After some experimentation, it was found that the best way to keep the stope full was to retreat the first sublevel back 50 feet or so and then start blasting the sublevel above. If the holes are in excess of 50 feet, then the bottom 25 feet or bottom half of the hole would be blasted for three or four rows ahead of the collars. Controlled pulling of the ore based on visual inspection is necessary for best results.

It depends entirely on the grade of ore and type of fill involved at the ends or above the stope as to whether or not an ore rib will be left to hold back the fill, if required. These ribs are established during the planning stage.

Vertical Crater Retreat

This method of blasting has been tested. The results indicated that in the Geco ground the 4½-inch holes would not allow a large enough spherical charge to make it an economical method.

Remnants

These are small, isolated blocks of ore and are generally surrounded by fill on three or more sides. Because of this, they are difficult to drill and load due to wall pressure as soon as any opening is developed. These blocks cannot be recovered utilizing 2½-inch [illegible] programs.

Experience in three of these remnants has confirmed that they can be recovered if drilled with 4½-inch-diameter downholes. A very tight pattern is required to assure fine muck, because the screens or drawpoints in these areas will not withstand secondary blasting. The drilling pattern used is generally a 5-foot zig-zag pattern.

Performance and Costs

Table 1 is a comparison of 4½-inch down-the-hole drilling performance in those stopes drilled during September and October 1980. The information is from the bonus sheets for the period stated.

Table 2 is a comparison of the 2½-inch drilling and loading costs with those for the 4½-inch holes. The costs are on a per-cubic-foot-broken basis. This type of calculation is required because of the difference between the average specific gravity of the ore in the western portion of the mine (3.0) and that in the eastern portion (2.8).

Conclusion

Geco is currently drilling 16,000 feet of 4½-inch-diameter blastholes per month with eight machines. Results from the blasting of the 4½-inch holes has confirmed the belief that savings were to be realized in the drilling, blasting and loading portion of the operation. Ore remnants previously thought to be non-recoverable are now extractable by the blasthole method. The 4½-inch holes have promoted different blasting procedures in the pillars and stopes, which has led to more stable wall and pillar conditions. The ability to drill a 4½-inch drain hole from level to level in one pass has decreased drilling costs for these holes by 50 per cent.

Present plans are to continue converting from 2½-inch to 4½-inch holes wherever possible. Because the high-pressure units have the ability to cut the drilling time of a stope or pillar by 50 per cent, it is planned to convert two more units, for a total of three, to high pressure.

Acknowledgments

The writer wishes to thank L.S. Brooks, mine manager, and P.C. McLeod, general mine superintendent, for permission to present and publish this paper and the staff for their help and suggestions in its preparation.

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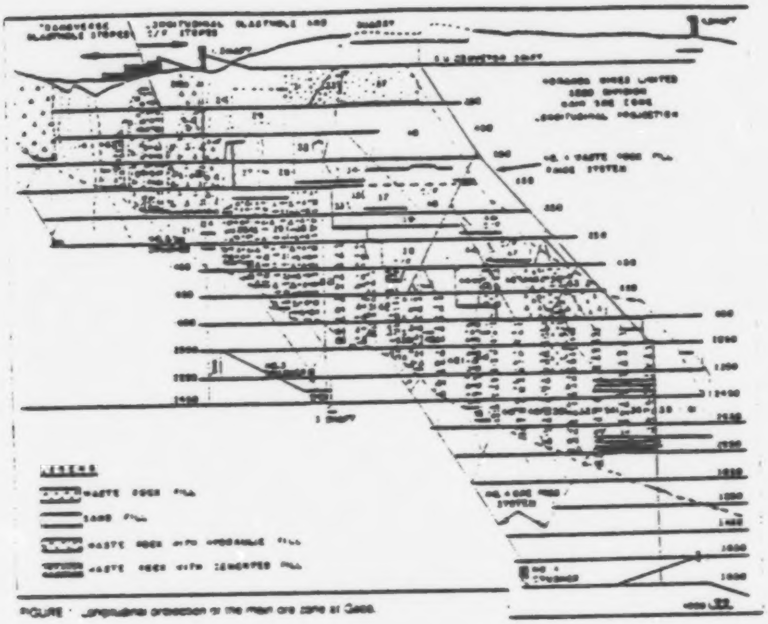
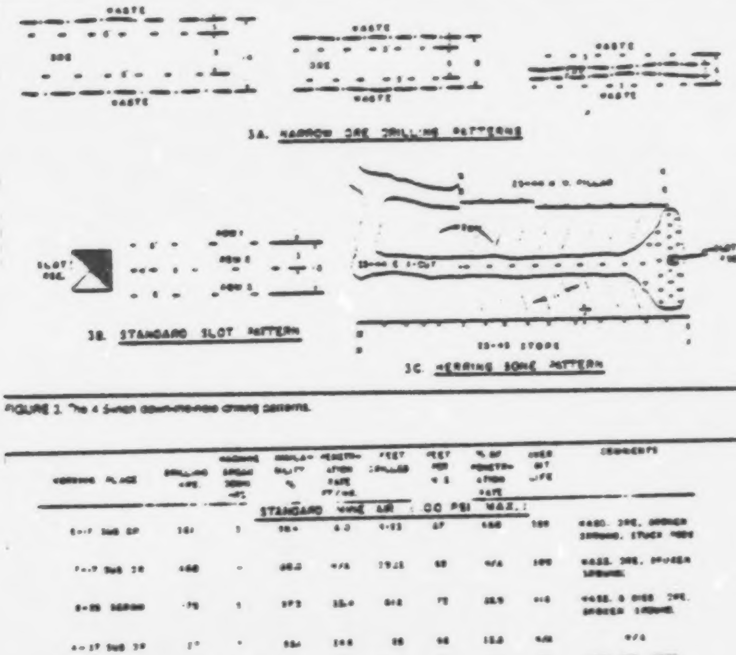


FIGURE 1. Longitudinal projection of the main ore zone at Geco.



3A. NARROW ORE DRILLING PATTERNS

3B. STANDARD SLOT PATTERN

3C. HERRING BONE PATTERN

FIGURE 3. The 4.5-inch down-the-hole drilling patterns.

WORKING PLACE	DRILLING HRS.	MACHINE BREAK DOWN HRS.	AVAILABILITY %	PENETRATION RATE FT/HR.	FEET DRILLED	FEET PER M.S.	% OF PENETRATION RATE	HOUR BIT LIFE	COMMENTS
			STANDARD MINE AIR (100 PSI MAX.)						
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	MASS. ORE, BROKEN GROUND, STUCK RODS
[illegible]	[illegible]	–	[illegible]	N/A	[illegible]	[illegible]	N/A	[illegible]	MASS. ORE, BROKEN GROUND.
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	MASS. & DISS. ORE, BROKEN GROUND.
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	N/A
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	MASS. ORE, GOOD GROUND.
[illegible]	[illegible]	[illegible]	[illegible]	N/A	[illegible]	[illegible]	N/A	[illegible]	MASS. ORE, GOOD GROUND.
[illegible]	[illegible]		[illegible]	N/A	[illegible]	[illegible]	[illegible]	[illegible]	DISS. ORE, GOOD GROUND HARD DRILLING
			HIGH PRESSURE (250 PSI MAX.)						
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	MASS. & DISS. ORE GOOD GROUND

TABLE 1. Down-the-hole drilling [illegible] performance for September and October [illegible]

MINING COSTS COST PER CUBIC FOOT BROKEN

		MINE AIR	MINE AIR	HIGH PRESSURE
DRILLING	LABOUR	[illegible]	[illegible]	[illegible]
	BITS	[illegible]	[illegible]	[illegible]
	DRILL RODS & MISC.	[illegible]	[illegible]	[illegible]
EQUIPMENT	MAINTE. & DEPRECIATION			
	ROCK DRILLS	[illegible]	[illegible]	[illegible]
	DRILL CARRIAGES	[illegible]	[illegible]	[illegible]
	BOOSTER COMPRESSOR	—	—	[illegible]
BLASTING	INCL. CLEANING HOLES			
	LABOUR	[illegible]	[illegible]	[illegible]
	SUPPLIES	[illegible]	[illegible]	[illegible]
TOTAL		[illegible]	[illegible]	[illegible]

TABLE 2. [illegible]

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CIM UNDERGROUND OPERATORS CONFERENCE

FEBRUARY 19-21, 1979

TIMMINS, ONT.

SESSION: LATERAL DEVELOPMENT

PAPER NO. 1

TITLE: MECHANIZATION IN LATERAL DEVELOPMENT

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ABSTRACT:

This paper reviews the increased use of mechanization in order to develop lateral headings efficiently and economically. Particular emphasis is placed on the role of the Electric-Hydraulic Drill Jumbo in achieving this goal. Also, the use of modified mining equipment to suit various operating conditions is discussed.

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During the 1970's, miners have constantly had to review their methods of mining in order to increase their productivity and reduce their operating costs in order to remain competitive. This paper discusses the role that the Electric Hydraulic Jumbo has played in achieving these goals.

With the introduction of electric hydraulic drills in the mid 1970's, the mining industry's approach to drilling was altered radically. Suddenly the drilling cycle time, which accounted for upwards of 60% of the time at the face, was cut in half. It was quickly found that the, mucking, bolting, and loading times had to be improved if the full advantages of the electric hydraulic drill were to be realized..

WHY ELECTRIC HYDRAULIC DRILLS?

The doubling of the penetration rates over air operated machines is obviously the most significiant reason and is justification alone for employing electric hydraulic drills.

The other major advantages of hydraulic drilling are:

(1) Better availability of the hydraulic drill because of fewer moving parts.

(2) Less maintenance on the drill booms because of less vibration from the drill.

(3) Quieter drilling.

(4) Better vision because of the elimination of fog.

(5) Much less electrical power required as compared to the equivalent drilling capacity with air operated drills.

(6) The hydraulic operation of the booms allows for smoother and finer control.

(7) The automation incorporated in hydraulic drills has reduced such problems as stuck steels, etc.

The disadvantages of hydraulic drills are:

(1) High capital cost.

(2) Shortage of highly trained hydraulic mechanics.

(3) Higher steel and bit costs.

(4) Difficulty in obtaining parts at times.

(5) A high voltage electrician is required on the job.

Two examples where the use of the electric hydraulic drills has resulted in a dramatic increase in performance are: one, in a single 17' x 16' drift heading and, the other in a 25' x 20' multiple drift heading situation. The main restriction in each example is that the blasting times are fixed at 8 hour intervals, thus all face activity must be scheduled according to blast time. Additional operating conditions are:

(1) The face and walls must be bolted on a 4' x 4' pattern prior to drilling.

(2) Perimeter blasting techniques must be employed.

(3) Haulage is 2000 ft. one way.

(4) Rock hardness is 30-40,000 psi.

SINGLE HEADING

Previously, employing a 3-boom air jumbo, two shifts were required to advance 10 ft. in a 17' x 16' heading. The cycle time and equipment is illustrated in Table I. Table II indicates the cycle time for the same heading employing an electric hydraulic drill.

In comparing Tables I and II, it can be seen that the electric hydraulic increased the performance by 33% and at the same time the face advance was doubled for a similar time period.

This better performance was achieved by employing an 8 yd scooptram and increasing the drilling capability of the rock-bolt jumbo. Without these changes the face cycle time could not have been reduced to less than 8 hours. Thus the full advantages gained from the electric drill were realized and the goal of driving a high-speed drift accomplished. However, in general, the electric hydraulic drill is not warranted for a single heading because of the high amount of idle time which results in high equipment costs per foot.

MULTIPLE HEADINGS

Previously, 25' x 20' drift headings were being driven using a 3 boom air drill to produce 400 tons per shift at a productivity of 67 tons/manshift. Table III summarizes the equipment crew, and cycle times. This existing operation was assessed with the goal being to double production while also increasing the productivity.

The immediate solution was to replace the 3-boom air drill jumbo with two 2-boom air drill jumbos. Thus the 15 hours of drill time which required 5 hours of face time for actual drilling was reduced to 3.75 hours of face time. The result was the ability to drill 600 tons per shift. Also the 5 yd scooptram was replaced by an 8 yd scooptram. A production rate of 600 tons/shift at a productivity of 85 tons/manshift was attained for the operation.

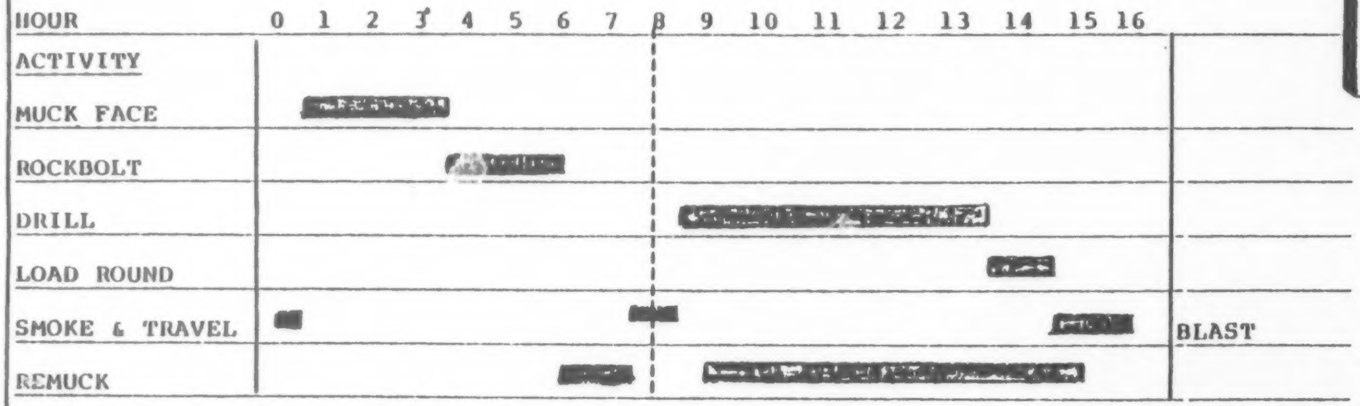
The next step was to introduce a 3-boom electric hydraulic drill once the electrical capacity had been installed. The hydraulic drill required a face time for drilling of 2.5 hours and thus was capable of completing two complete 400 ton rounds per shift. Table IV indicates the cycle times for this situation. The goal of doubling production had been achieved. However, the desired increase in productivity could not have been attained, if the existing haulage equipment was merely doubled to handle the higher tonnage. Doubling the haulage equipment resulted in a productivity of only 89 tons/manshift. After reviewing the haulage equipment and because of the large heading size available, a Caterpillar 988 front-end loader was selected for mucking. At the same time, a 40 ton truck was designed and built to complement the mining operation. The result of the combination of the hydraulic drill, 988 loader, and 40 ton truck was a production rate of 800 tons per shift at a productivity of 114 tons/manshift.

The preceding examples illustrate that when matched with the proper equipment for mucking and bolting, the electric hydraulic drill can produce dramatic increases in performance. But the other key item to achieving and maintaining these performances is to ensure that the service equipment is also as efficient. For example, proper man carriers, jeeps, and utility trucks must back up this equipment. Mechanics must be equally as mobile and fully equipped to provide the same performance. Such items as scissor lifts must be used to install services, for unless this part of the operation is as efficient, the full cost benefits of the face equipment are not gained.

CONCLUSIONS

While the introduction of the electric hydraulic jumbo has produced significant increases in performance, the selection of the support equipment from mucking to mechanics transportation is just as important in attaining the high overall performance and hence a reduced operating cost. As of today, the muck removal has not kept pace with the increased drilling capabilities, and is, in our opinion, a most critical area that has to be solved if the full impact of faster penetration is to be realized.

SINGLE HEADING 17' x 16'

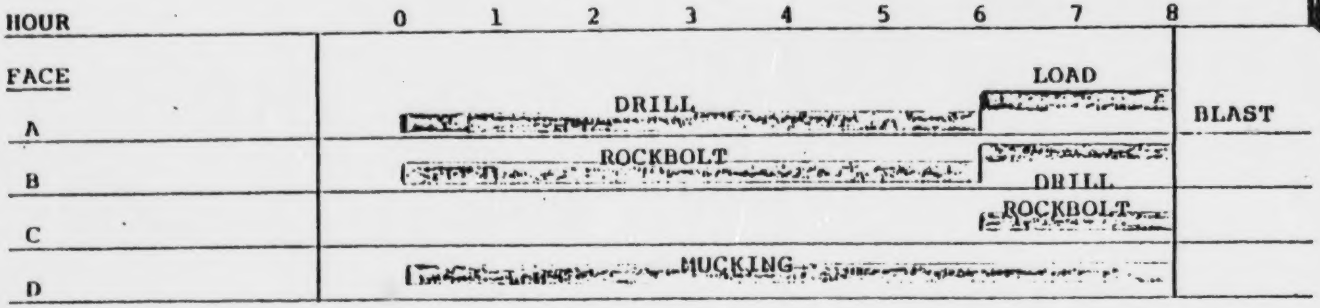


EQUIPMENT	CREW	
1 - 5 yd. Scooptram	1 - Scoop/Rockbolter	TOTAL BREAK - 10'
1 - Rockbolt Jumbo	1 - Scooptram Operator	TOTAL MANSHIFTS - 4
1 - Loading Jumbo	1 - Driller	PERFORMANCE - 2.5 ft/ms
1 - 3-Boom Air Drill Jumbo	1 - Faceman	3 SHIFT ADVANCE - 15 ft.

3-BOOM AIR DRILL JUMBO - SINGLE HEADINGS

TABLE I

MULTIPLE HEADING - (25' x 20')

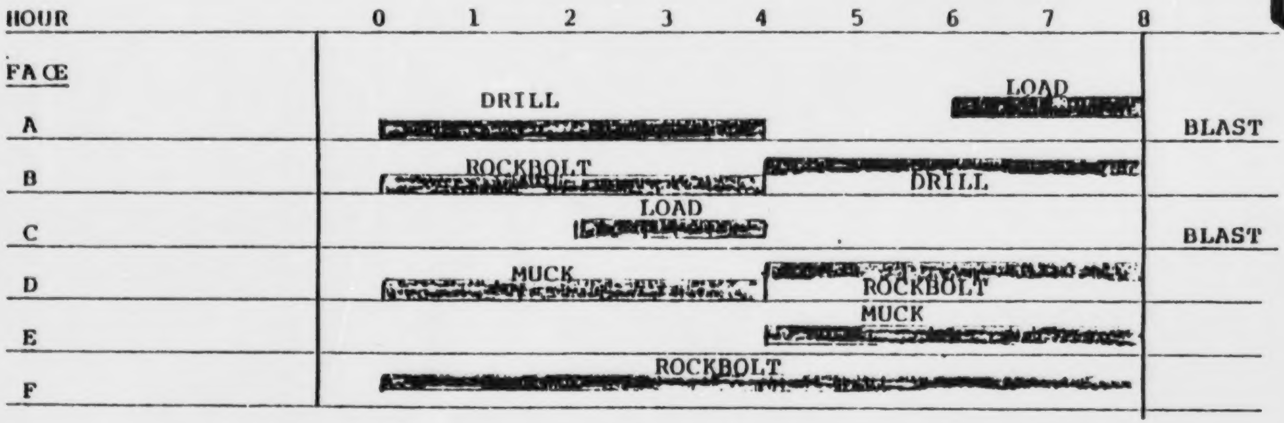


EQUIPMENT	CREW	
1- 3-Boom Air Drill Jumbo	1 - Rockbolter	TOTAL BREAK - 400 tons
1 - Rockbolt Jumbo	1 - Driller	
1 - Loading Jumbo	1 - Faceman	TOTAL MANSHIFTS - 6
1 - 5 yd Scooptram	1 - Truck Operator	
1 - 20 Ton Truck	1 - Scoop Operator	PRODUCTIVITY - 67 tons/ms
	1 - Loading Jumbo Op	

3-BOOM AIR JUMBO - MULTIPLE HEADING

TABLE III

MULTIPLE HEADINGS - (25'x20')



EQUIPMENT	CREW
1 - 3-Boom E.H. Jumbo	2 - Rockbolters
2 - Rockbolt Jumbo	1 - Driller
1 - Loading Jumbo	1 - Faceman
1 - Caterpillar 988 Loader	1 - Truck Operator
1 - 40 Ton Truck	1 - 988 Operator
	1 - Loading Jumbo Op

TOTAL BREAK - 800 tons

TOTAL MANSHIFTS - 7

PRODUCTIVITY- 114tons/ms

3-BOOM ELECTRIC HYDRAULIC DRILL - MULTIPLE HEADING
TABLE IV

by T. L. LAKANEN
Mine Superintendent, Lockerby Mine, Falconbridge Nickel Mines Ltd.

Remote Control Load-Haul-Dump Units

The Lockerby mine is the newest of five producing mines operated by Falconbridge Nickel Mines Ltd. in the Sudbury District of Ontario. It will be into full production early this year. Eighty-two percent of the mine's production will come from large diameter blast hole stoping with the remainder, covering the fringe areas, from a tight rill (Avoca) mining method and some conventional cut-and-fill stoping. This article deals with the introduction, experience and the impact of remote-controlled load-haul-dump equipment used in conjunction with the large diameter blast hole stoping as practiced at the Lockerby mine.

Ore body is steep dipping

To get a clearer picture regarding the setting and the actual use of load-haul-dump equipment at Lockerby, a look at the geology and geometry of the ore with a general understanding of the level operation and the mining method is necessary (see fig. 1-3).

Regarding the ore body depicted in fig. 1, the points to note are as follows:

- Primarily steep dipping and conforms to the shape of a bow.
- The inside area of the bow (south side) is relatively competent granite and harbors all accesses to the ore.
- The outside area of the bow (north side) is a rather blocky, incompetent norite.
- One transverse blast hole block extends from the 3400-level horizon up to the 2600-level horizon and the second from the 2200-level horizon to the 1700 level.
- Ore widths in the transverse areas vary from 40 to 180 ft with an average of about 74 ft. These ore widths translate into stope lengths for transverse mining.

Of interest in fig. 2 are:

- The level intervals are 175 ft.
- The ore along strike is divided into 36-ft wide panels.

This then provides transverse stopes that are 175 ft high by 36 ft wide by a length which is the thickness of the ore at that location (it varies from 40 to 180 ft). It is important to note that in relation to general blast hole concepts, overall stope sizes are very small. The stope width selected was dictated by the competency of the wall rocks and the mining depth.

In regard to general level layout, as represented in fig. 3:

- Ore and waste pass dumps are centrally located along strike.
- Each stope has only one access per level which serves as the entrance for: (1) Developing the undercuts and overcuts. (2) Servicing the stope below the level, and (3) the entrance for muck removal for the stope above the level.
- The stopes are short in relation to general blast hole mining concepts. Because recovery is related to stope length, (i.e., the shorter the stope the higher the overall recovery), it is this short length that makes for the fairly high recoveries even without remote controlled units. This point becomes more apparent later on.

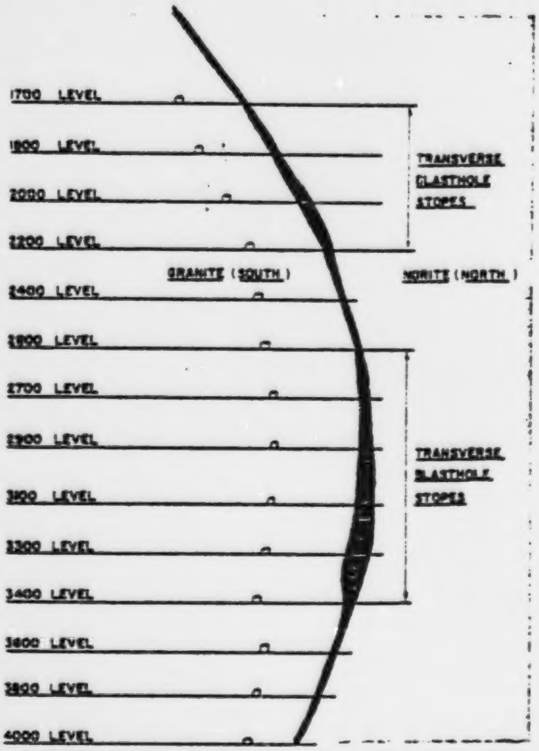


Fig. 1. Typical section through ore body at the Lockerby mine, Sudbury District, Ontario

This simple level plan, which satisfied the blast hole needs, required the least overall development of all the proposals investigated. This fact is of utmost significance because it results in earlier revenues at a faster rate than any of the others.

Points of interest in fig. 4 include:

- Cemented cable bolt support at both ends of the stope to reduce short term sloughing and to assist in long term ground support for future pillar mining.
- Installation of 10-ft cemented rebars in the back of

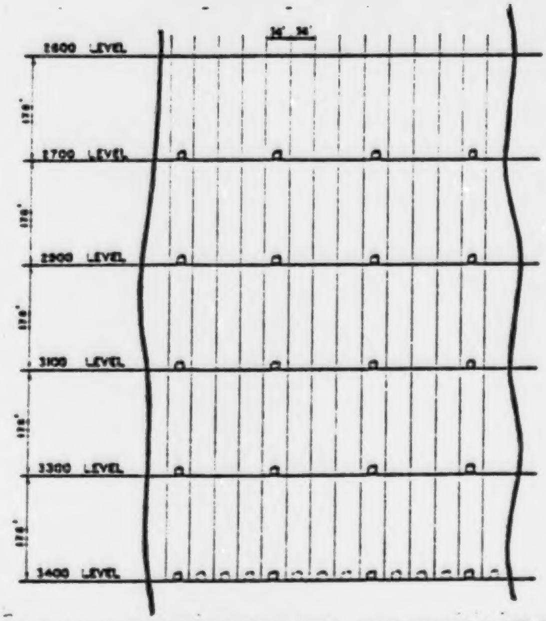


Fig. 2. Longitudinal section along strike. Level interval is 175 ft and stopes are 36 ft wide and as long as the thickness of ore at a given location

- the overcut along with total screening and shotcreting. This precaution was considered necessary due to the fact that the present overcut would later become the undercut for the stope above.
- Not evident in fig. 4 but, along with decoupled blasting of all pillar holes, additional holes on the pillars were part of the stoping plan to provide control and smoother walls.
- The mining starts at the far end and retreats towards the entrance.

The first three of the above four points were all incorporated in the stoping plan and are mentioned here because they are significant with regard to the safety of remote equipment in regard to falls of ground. The retreat blast hole face itself remains as the only serious source of such falls.

New mining concepts evolved during mine development

Initially the mine had been planned for transverse open cut-and-fill stoping. During the development stages, new concepts were developed so numerous comparative feasibility studies were run considering different methods and the development requirements associated with each. The transverse large diameter blast hole method was the most attractive if 87 percent recovery of the total ore could be assured. Even with one entry, our studies showed this could be achieved.

An examination of duplicating the level development to provide a second entrance at the other end of the stope indicated an increased recovery of ore (approximately 92 percent) and made this more attractive than the single drawpoint. However, earlier work had shown that shorter stopes produced higher recoveries, so this avenue was pursued. This study showed that with one entrance and by mining the longer stopes in parts, as high or higher recoveries than the two entrance syste could be achieved at less overall expense and also with earlier production returns.

The final decision was to go ahead with only one e trance per stope and stopes longer than 60 ft would mined in two equal parts while stopes larger than 90 ft would be taken in three equal parts. This criteria indicated a blast hole recovery in the transverse stopi areas of 93 percent.

However, with the introduction of remote controlled equipment, all these foregoing considerations and problems were eliminated. Intermitting would no longer necessary although it would require periodic cleani out of a stope before all breaking was completed to avoid oxidation. The most important benefit, however, was that now full recovery of what was broken could safely accomplished.

Remote mucking tested in two stopes

Phase 1. Incorporated, as priority, in the Lockerb schedule was the development and mining of two blast hole stopes for the purpose of testing the method. In t first stope, after 90 percent of the broken ore had be extracted by conventional LHD operation under the safety of the brow, a mucking machine with remote controls was utilized to clean out the remainder of the sto for full recovery. This procedure was repeated in t second test stope with the same results. The mining method had proven up and the operators had shown that full recovery was possible. However, some bas disadvantages were recognized:

- The mucking machine was too small and under-

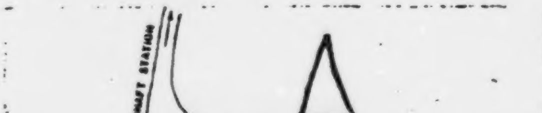


Fig. 3. General level plan at the Lockerby mine. This plan required the least overall development of all proposals investigated

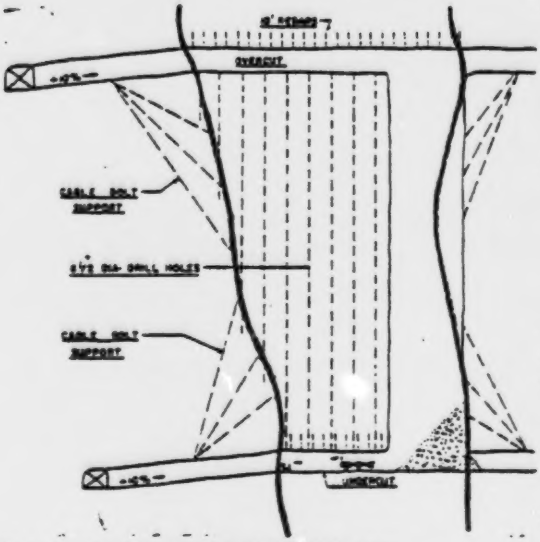


Fig. 4. Section through a typical blast hole stope

powered for the height of the muckpiles, which were approximately 70 ft high. This resulted in relatively high maintenance costs.

- Power for the mucker and the remote signals were transmitted through a trailing air hose and power cable that were strapped together. Occasionally the machine would run over its own air hose causing it to stall in the open stope. This had been anticipated and a steel cable had been secured to the back of the machine with the other end snubbed to I-bolts inside the entrance. Thus it was possible for the stalled machine to be recovered. Although the cable served its purpose, it also tended to get tangled in the wheels of the mucking machine.
- Due to this air and cable connection and the distance to the orepass dump, it was necessary to provide a dump point where the material was picked up by LHD and taken to the orepass.

The remote mucker, however, proved to be a very useful piece of equipment and is still utilized at Lockerby mine for remote mucking of smaller areas on levels where remote LHDs are not justified. It is also used in conjunction with remote, radio-controlled scoops for two purposes. One is for placing charges for secondary blasting and the second is for rescue operations of stalled LHDs. The last point will be mentioned in more detail later on. The main advantage of the mucking machine is that it fits into the service cage and can be readily moved between levels.

Phase 2. At this point, remote mucking had been proven both safe and economical. Mine Engineering Services then investigated the development of a remote controlled LHD preferably without any trailing connections. It was known that Cobar Mines Pty. Ltd. in New South Wales, Australia had successfully developed and used a remote radio-controlled system for a Wagner ST-5 scoop and they provided information on their unit.

The Instrumentation Development Group from Falconbridge worked in conjunction with an LHD manufacturer on this project. They designed the present sending and receiving units and in cooperation with Jarvis Clark Ltd. of North Bay, Ont., developed the hydraulic and pneumatic interface hardware. Initial testing of the system was carried out in North Bay (fig. 5) and in June 1977, a JS-500 scoop equipped with radio controls was brought on site for testing underground at Lockerby mine. This unit successfully mucked out three stopes for full recovery, and five units equipped for remote operations were ordered for installation in the mine by the end of 1979.

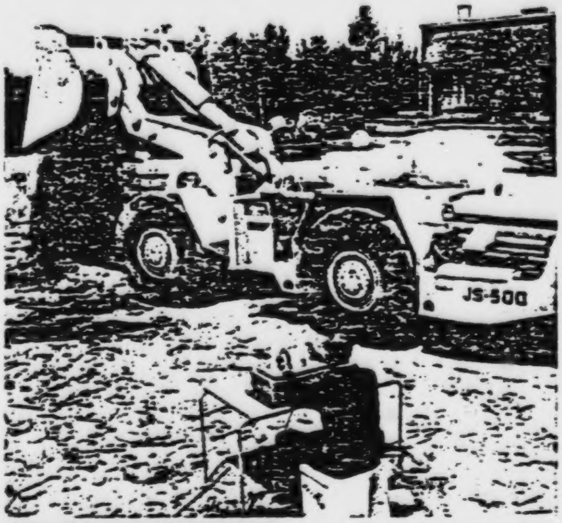


Fig. 5. After tests of the remote-control system on surface, a scoop with radio controls was placed underground at the Lockerby mine for additional testing. Unit successfully mucked out three stopes

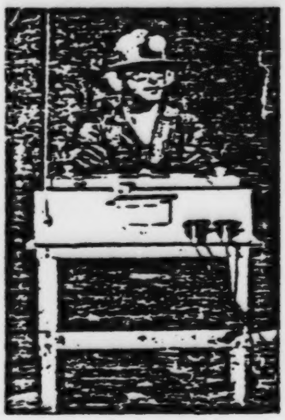


Fig. 6. Operator's remote radio console duplicates functions in LHD cockpit as closely as possible. Console is on a suspended staging to provide the best possible visibility of machine operating area in stope

Scoop has numerous safety features

For obvious reasons, the console simulates the actual LHD operator's cockpit as closely as possible, including foot pedals (fig. 6). Safety features included in the system are as follows:

- To insure against initiating an electric blast, actual tests verified that radio frequency radiation was within the government standards.
- The radio control frequency system (the same as used to control model airplaines) was tested in an underground environment to ensure that no possibility of frequency interference from outside sources existed.
- Safety against a runaway situation:
  - —An emergency brake button on the console is

readily available for stopping the scoop quickly at any time.

—There is also an emergency stop button on the console which not only shuts off the motor but at the same time kills the radio signal.

—A carrier control in the receiver will, upon loss of the signal, automatically apply the brakes, reduce the throttle down to idle and kill all other vehicle functions.

—A "dead-man" control on the console throttle pedal insures that when the foot is lifted the scoop throttles to idle and the brakes set automatically.

—To operate in the manual phase, the transmitter must be shut off before the scoop will function.

Machine operates on manual control to orepass

The operator's stand is located on a suspended staging in the safety of the stope entrance. This location provides the best visibility of the entire machine area. The staging has sufficient clearance for safe passage of the scoop underneath it. The operator pilots the scoop from this console position into the open stope, fills the bucket and recalls the loader into the protection afforded at the drawpoint entrance. From this point, the operator leaves the console, mounts the scoop and resets the controls for manual operation to run the scoop out to the orepass dump. Upon return to the drawpoint, the operator once again sets the machine on remote and takes over operation from the console.

Signals emitted from the console transmitter are picked up by a receiver mounted on the scoop and are translated into machine actions by the hydraulic and pneumatic interface hardware. The basic system components are relatively simple in composition and have proven durable enough to withstand the constant heavy workload to which mining machines are subjected.

To improve visibility, additional lights were mounted on the LHD. In addition, stope lights were installed at both undercut and overcut entrances. This provides adequate lighting for mucking up to 150 ft from the operator's position.

Provisions made for rescue of stalled machine

The risk of possible loss of the LHD due to falls of ground was a serious factor that had to be considered. However, at Lockerby mine, this risk was considerably reduced by precautions and preparations that were already an integral part of the mining scheme as pointed out in the description of fig. 4. The risk, however, still existed and provisions to rescue a stalled machine are as follows:

- A scoop stalled in the articulated position would be difficult to pull out. To reduce the chance of this occurring, the operators were instructed to always travel from the entrance to the point of mucking in a straight line and avoid articulating the machine as much as possible.
- A hook designed to accept a looped tow cable was installed on the back end of the scoop. Unlike the mucking machine, a permanent snubbed trailing cable was not considered practical because of the travel to the orepass. The tow cable itself would be carried to the stalled machine using a remote controlled mucking machine.
- To release the automatic emergency brake in a sta situation, an essential part of the recovery system included a means of releasing this brake when a pull was exerted to the hook.

We did have an occasion to test our rescue operation when the machine was damaged by a fall of ground from the face. This fall was due to a condition that was created when only part of a production line was blaste resulting in an irregular face. In this incident, the remote mucking machine mucked the scoop clear, hooked the tow rope to the scoop and with the aid of another LHD vehicle towed the damaged machine to the acces drift.

Remote control concept highly successful at Lockerby

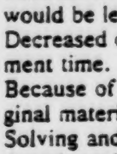
Operator acceptance of the radio-controlled JS-500 Scooptram was immediate, and it was found that an individual could be trained to use the remote control within one eight-hour shift. The average length of time for mucking cycle, within which broken ore is loaded remotely from a pile 100 ft from the operator and then shuttled in the conventional manner to a dump point 10 yd from the entrance of the stope, is less than five minutes.

The implementation of the remote control concept at Lockerby mine is considered to be highly successfu This feeling is substantiated by the following summar of the impact it had on this mining operation:

- Increased recoverable reserves with expected increase in the life of the mine.
- Substantial increase in revenue because ore tha would be left now returns an immediate revenue.
- Decreased development expenditures and development time.
- Because of the three preceeding items, some marginal material becomes ore.
- Solving and simplifying some mining problems reduced stoping costs.
- Reduced employee exposure to potential hazards.

When the impact of all of these, or each one in itself, is considered against the investment cost of adding remote unit to an existing piece of equipment, it can b seen that the return of this investment can be exceptional.

A mine in Ireland is at present using a radio-remote controlled unit developed by Falconbridge personne At the Sudbury operations, the Mine Engineering Services Group is evaluating the possibilities of adding remote controls to equipment at other operating properties. It is believed that future mining methods will b developed and designed providing greater application of this new tool. ◆



T. L. Lakanen is in his 27th year wit Falconbridge Nickel Mines Ltd. He is currently superintendent of the Lockerby mine, which has been converte recently to bulk mining methods usin large diameter blast holes from cut-and-fill mining. Lakanen's experience with the company has included supervisory positions at several undergroun properties during development, and h has also worked as area research engineer, mine engineer and mine planning engineer.

MAJOR TECHNOLOGICAL ADVANCES
IN THE
DESIGN
OF
TRACKLESS MINING EQUIPMENT
IN
THE 70'S

Paper to be presented at the 82nd Annual Meeting
of the C.I.M. (Toronto, April 22, 1980)

By: R.A. Richard
V.P. Marketing
Jarvis Clark Co. Ltd.

Several technological breakthroughs in the design of trackless equipment have benefitted the underground mining industry in the last decade. This paper will be concerned with major innovations which have resulted in higher safety, availability or productivity and in some cases the solution to specific problems which resulted in high maintenance since the introduction of trackless equipment in Canada during the mid sixties.

THE ELECTRIC LHD

Perhaps the most significant of all changes was the introduction of the electric scoop in the early 70's. It took a while for the industry to recognize the potential cost savings generated by these units mainly because energy savings were not always taken into consideration when comparing the electric LHD to the diesel. In addition there were some componentry problems related to the hydrostatic systems used for speed variation and generally a lack of recognition of the requirement for education of the operator in coping with the electric cable.

Nevertheless, there were some success stories as shown below in Table I which illustrates the results of the first twenty months of operation with the prototype JS-100E delivered to Dome Mines in Timmins in January 1973.

TABLE I

TOTAL TONS MUCKED	114,589
TOTAL HOURS IN WORKING PLACE	5,375
AVERAGE TRAMMING DISTANCE	165 FT.
PERCENTAGE UTILIZATION	62%
HOURS ACTUALLY WORKED	3,300
TONS PER HOUR	35
AVAILABILITY	95.4%
UNSERVICEABLE TIME	4.6%

These encouraging results kept us as a manufacturer encouraged in the potential of the electric LHD and the 2.2 cubic yard capacity was introduced in 1974. This design also used hydrostatic drive for speed variation and some problems were also encountered which appeared to be related to contamination of the hydraulic oil by dirt, condensation water and also shock loading which damaged the swash plate in the hydrostatic pump.

The separation of the hydraulic system from the hydrostatic appears to have resolved most of the problem, i.e. contamination, but hydrostatic systems do appear to be sensitive to powertrain shock loading such as experienced in LHD during the loading part of the cycle. For this reason a new approach was used when the electric 5 cubic yard LHD was designed.

In the JS-500E the powertrain components are the same as used in the diesel version, i.e. torque converter and power shift transmission. Speed variation is achieved through the use of a totally enclosed tube air cooled DC motor with conversion of the AC power source to DC being achieved on board the unit by an SCR converter. The big concern at design time was the life expectancy of solid state components on board a unit which is subject to shocks generated by ramming the muck pile and continuous pounding against solid rock walls.

After three years of experience we are now of the opinion that this system is by far superior to hydrostatic componentry. Since no problems have been experienced with the SCR converter, this system would immediately be introduced in the smaller LHD if the problem of lack of physical space in the envelope of these units would be resolved. However, similar systems to the JS-500E are being contemplated for the 3.5 cubic yard and 8 cubic yard LHD.

Despite componentry and adaption problems, the electric LHD is here to stay and I predict we will see a more widespread application in years to come to the point where mining methods will be developed around the limitations of the umbilical cord. Recent visits to countries like Sweden, South Africa, Rhodesia, and Zambia have confirmed my confidence in the electrically powered LHD which started with the development of the first unit in the early 70's. The trends do support my prediction. These are shown in Table II below.

ELECTRIC UNITS SOLD BY JARVIS CLARK

1973 - 1
1974 - 5
1975 - 11
1976 - 17
1977 - 23
1978 - 24
1979 - 29

The justification for going to electric LHD's is different in the various countries but all are related to economical use of energy.

In the Northern hemisphere the two main factors are generally the cost of heating the ventilating air and the cost of fossile fuel versus the use of the more efficient electrical power which is ample justification for their future use. In Canada it is generally recognized that ventilation costs average $2.50 per CFM per year. When one considers that diesels require on the average 100 CFM of ventilation air per horsepower, the electrics do make sense. These ventilation cost figures were confirmed by operators in Sweden in early 1979.

In the mines in the Southern hemisphere, particularly in the countries which do not have oil as a natural resource, but ample electrical power, the interest in the electrics has been considerable. In South Africa for instance, a gallon of diesel fuel costs the equivalent of $2.90 whereby a KWH costs the equivalent of $.022. The net saving per year is in the order of $45,000 for a 5 cubic yard unit.

Also, in many of the deep mines they have to refrigerate the ventilating air at a cost equivalent to that experienced in Canada to heat it. Hence the electrics which produce less heat than diesel does make a lot of sense.

In addition to energy use consideration, the electrics offer several advantages over the diesel. These are:-

- Lower noise level
- Less dust and particulate matter in the underground environment
- Improved tire life
- Higher breakout since one can overdraw on the electric motor
- Reduced maintenance

RADIO REMOTE CONTROL LHD

The incentive for the development of "remote control" LHD equipment was to improve both productivity and safety in certain mining applications. A good example of this development work is seen in a paper by J.T. Brady.(2) This Australian author describes early experimentation conducted with an umbical cord remote control which later progressed to a radio controlled unit.

The mining method was a regular cut and fill stope application. Concern had been expressed over delays in productivity and the hazards of working under a most unstable back. It was reasoned that the occasional small piece of loose while potentially fatal to an operator could be accepted by a remotely controlled LHD.

The trailing cable or umbilical cord concept was tried and rejected because of a number of limitations. Specifically, restricted tramming distance, problem with retrieval and difficulties experienced in manual operation of the LHD for part of the cycle. A radio controlled unit was built and is operating quite satisfactorily.

In addition to the above radio remote controlled LHD, several have been used in Canada for the past three years in a variety of hazardous applications. In one mining method, the blasted muck in an unprotected open stope is being recovered using a radio remote controlled JS-500. Without this machine a percentage of the ore could not be recovered. Within two months the operators have claimed that the additional tons of ore removed are worth approximately three times the costs of the RRC scoop.

Some of the design considerations in this development necessarily revolved around the safety of such a system, i.e.

- low radio power output of transmitter to eliminate hazards of setting off electric detonator.
- matched radio sets which can identify their own signals only.
- control box which is portable so that operator is in line or sight with the unit.
- control box design which can be handled easily by a regular scoop operator.

These basic design criteria were met as follows:-

TRANSMITTER RECEIVER SELECTION

A search of the marketplace was made for commercially available radio sets with a sufficient number of channels to handle the various functions required to remotely operate the unit. These include:-

- engine throttle
- service braking
- emergency braking
- boom control
- bucket control
- steering
- engine start
- deadman

The radio systems had to be built to comply with the various CSA codes, and are licensed for operation by the Federal Department of Communication. The power output is less than one watt, and is incapable of inducing a current in blasting circuit which could detonate blasting caps.

Because of the possibility of operating several of these units within a defined mining area and also the safety hazards possible if the receiver could pick up radio waves from other sources, the transmitted signals are usually frequency-modulated, using phase-shift or frequency-shift techniques. The modulating signal is binary-coded data. The operator's remote control signals are scanned 20 to 30 times each second within the transmitter, and the resulting data is transmitted in messages to the machine-mounted receiver. The message is a binary pulse

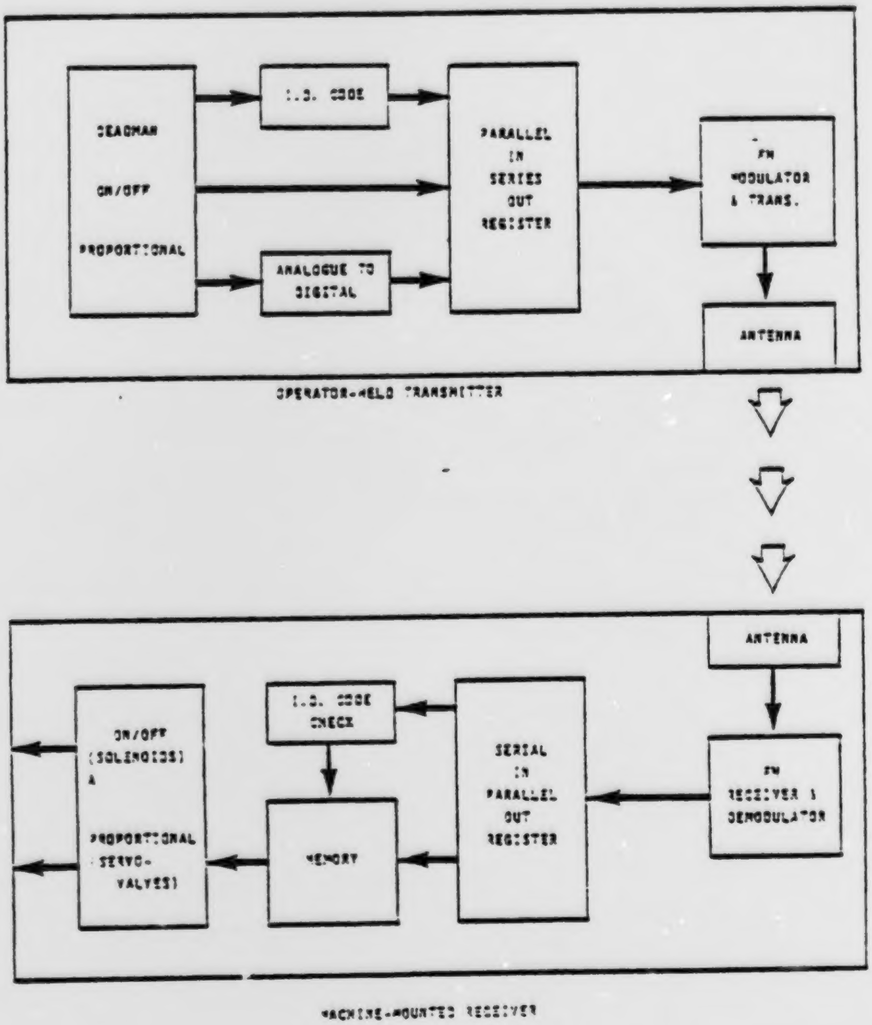

DEADMAN
ON/OFF
PROPORTIONAL
I.D. CODE
ANALOGUE TO DIGITAL
PARALLEL IN SERIES OUT REGISTER
FM MODULATOR & TRANS.
ANTENNA
OPERATOR-HELD TRANSMITTER
ANTENNA
FM RECEIVER & DEMODULATOR
SERIAL IN PARALLEL OUT REGISTER
I.D. CODE CHECK
MEMORY
ON/OFF (SOLENOIDS) & PROPORTIONAL (SERVO-VALVES)
MACHINE-MOUNTED RECEIVER

FIGURE 1

train consisting of timing, identification and control data. Each manufacturer of radio control equipment has evolved his own specific coding which has been designed to minimize the possibility of responses to spurious signals.

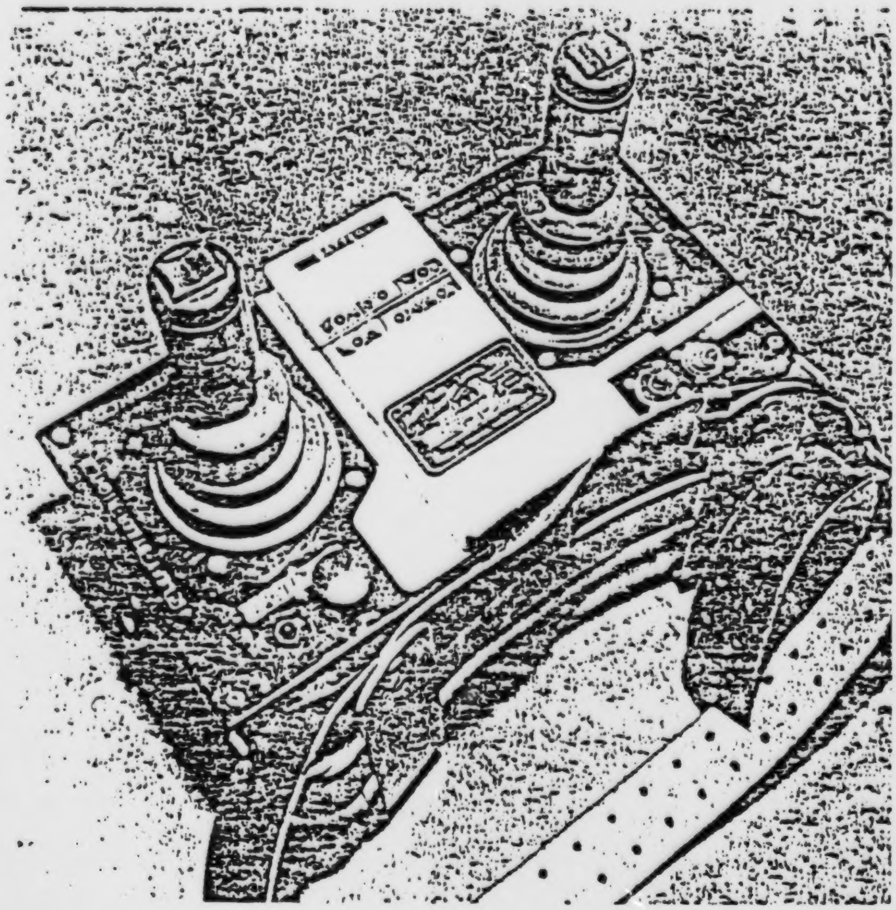
Figure 3 shows a typical transmitter and receiver system.

CONTROLS

The ergonomic challenge of hand operation required close consultation with experienced load-haul-dump operators. These miners know which combination of manoeuvers are necessary to successfully load a bucket, and we are grateful for their help and advice in our development.

A successful design of operator's console is shown in Figure 4. Here we have two joystick levers with four quadrant operation and a combination of on/off switches. The console is battery powered using non-rechargeable mercury dry cells which have proven to be more popular than either the headlamp wet cell or nickel cadmium alternatives.

The machine mounted receiver is designed to compliment the transmitter in its capability of translating the messages into on/off and proportional signal outputs.



The safety aspect of spurious radio interference is overcome by such design techniques as:-

a) receiver processes only data on its designated frequency.

b) both receiver and transmitter have matching ID codes to restrict operation to designated pairs

c) the received data configuration must meet a predetermined pattern of ID, timing and data pulses. In some systems the data must be valid for three or more consecutive messages before an output will be generated. This effectively controls random noise and interference between the transmitter and receiver.

d) the receiver output will return to zero if valid data is not received for half a second.

e) proportional signals also have limited ramp rates to guard against unexpected actions.

On loss of valid radio control, the machine is arranged to "fail safe", i.e transmission in neutral, brakes applied, etc. To avoid nuisance shutdowns, the engine or electric motor can be kept running or shutdown after a time delay.

Based on three years of operation of these units in Canadian mines, we can report that radio remote control systems are viable in the underground mining environment, and that the marriage of this technology to conventional load-haul-dump equipment has progressed with surprisingly few development headaches. Furthermore, we predict that this newly developed technology could lead to the development of mining methods which will feature improved safety and productivity.

As an example, and in combination with the VCR method of blasting, silling on the level is possible; no sub level would be required which would eliminate a considerable amount of development.

IMPROVED BRAKING SYSTEMS

The experience in the sixties where some runaways of trackless mining equipment occurred made both operators and designers aware of the paramount importance of safe braking systems, particularly with units weighing in some cases upwards of 80 tons and operating on decline gradients as high as 30%.

Runaway accidents investigated by our company showed in all cases that first generation designs were in fact adequate; also that runaways were in most cases the result of either inadequate maintenance of brakes or poor understanding of the braking system by either maintenance or operating personnel. This in turn indicated to us as equipment designers that additional safety factors had to be provided in the design along with back-up systems which would be automatically applied in case of an emergency. To achieve this desireable aim, we have been conducting a continuous component search for braking mechanisms that would reduce maintenance frequency realizing that the mining industry was plagued, and still is, by a shortage of heavy equipment mechanics.

The design changes introduced in the 70's could fall into three main categories:-

- actuating systems
- service brake mechanism
- back-up systems

The first step in the improvement of the straight air actuating system was the introduction of independent circuit for each axle along with circuitry which allowed each axle to be tested individually. The major limitation of this system is that braking torque was dependent on air pressure in reservoir stored in the unit. Air leaks and repeated brake applications could lead to brake fading.

To alleviate the problems inherent to the straight air system the air over hydraulic actuating systems were introduced in the early seventies. The main advantage was constant braking torque which could be set through hydraulic pressures. With this design improvement, along with the selection of proper componentry we decided that improvement in safety margin could be achieved and we adopted the following design criteria:

> "each axle system should alone be capable of stopping the unit fully loaded on a 25% gradient."

Along with this new design, we introduced the "maxi can", or lock-on actuator which cannot be released until a safe operating pressure is reached in the air system, usually set a 85 to 90 psi. Figure 5 shows the schematic of this system.

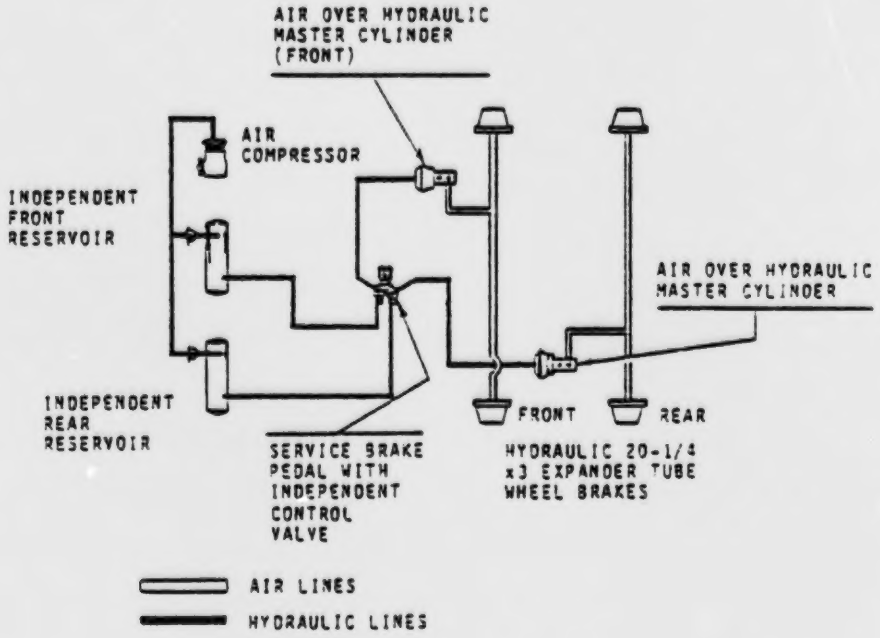

AIR OVER HYDRAULIC
MASTER CYLINDER
(FRONT)
AIR
COMPRESSOR
INDEPENDENT
FRONT
RESERVOIR
AIR OVER HYDRAULIC
MASTER CYLINDER
INDEPENDENT
REAR
RESERVOIR
FRONT
REAR
SERVICE BRAKE
PEDAL WITH
INDEPENDENT
CONTROL
VALVE
HYDRAULIC 20-1/4
x3 EXPANDER TUBE
WHEEL BRAKES
AIR LINES
HYDRAULIC LINES

FIGURE 5

During the same period as we were developing improved actuating system we were continuously on the lookout for improved braking mechanisms and at the same time putting pressure on brake component manufacturers to provide us with componentry which would improve brake maintenance interval. A study made in one mine in 1974 showed that an 8 cubic yard LHD equipped with expander tube brakes had an average tube life of 280 hours.

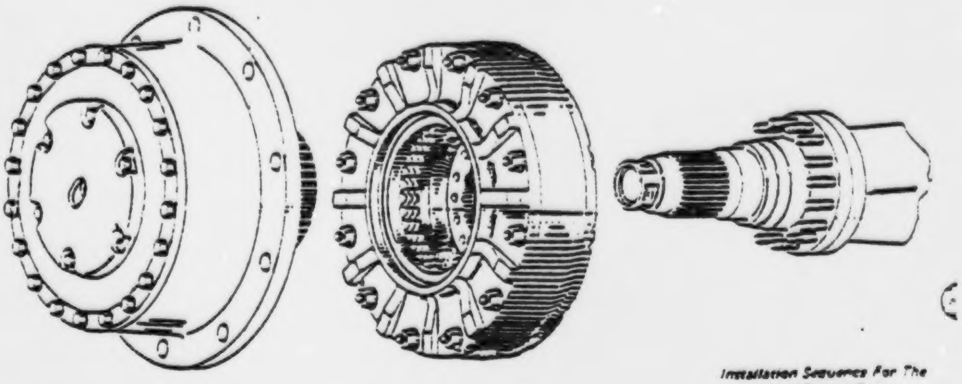
It should be mentioned that these units were operating mostly on declines with the front axle in murky water when mucking out a round. The first breakthrough in improved braking mechanisms came with the availability of dry disc brakes in 1974. These are now standard on most common models of trackless mining equipment. Maintenance intervals have been considerably improved to a range of 900-1500 hours. Along with this improved life there is greater ease of servicing because of external access to the brake pads.

However, we feel that the ultimate answer lies in the use of "totally enclosed wet disc brakes". These were introduced in the smaller LHD units in the late seventies and have been in use in some surface equipment for 4 to 5 years. The manufacturers recommended maintenance interval for these units varies between 7,500 and 10,000 hours.

The availability of these was announced by Rockwell in 1978 and by Clark in 1979. The system is described as "hydraulically actuated oil immersed multi plate liquid cooled friction disc brake."

Braking action is achieved through the application of the hydraulic piston with the rotating graphite friction surfaces which react with stationary stator plates retained by scalloped tongs on the outer diameter. These scalloped tongs transfer the reaction torque to the rigid outside housing. This new braking system is illustrated in Figure 6.

Figure 6.



Installation Sequence For The Liquid Cooled Brake

This improved design will be available for LHD up to 3.5 cubic yard capacity in 1980 and larger units in 1981. It is also designed in such a way that retrofitting of some of the older axles will be possible.

The other major area of brake improvement which took place in the early 70's was the provision of a back-up system. We felt that in the event of service brake failure our design should provide the ultimate, i.e. a fail safe spring applied hydraulically released brake with automatic application in case of loss

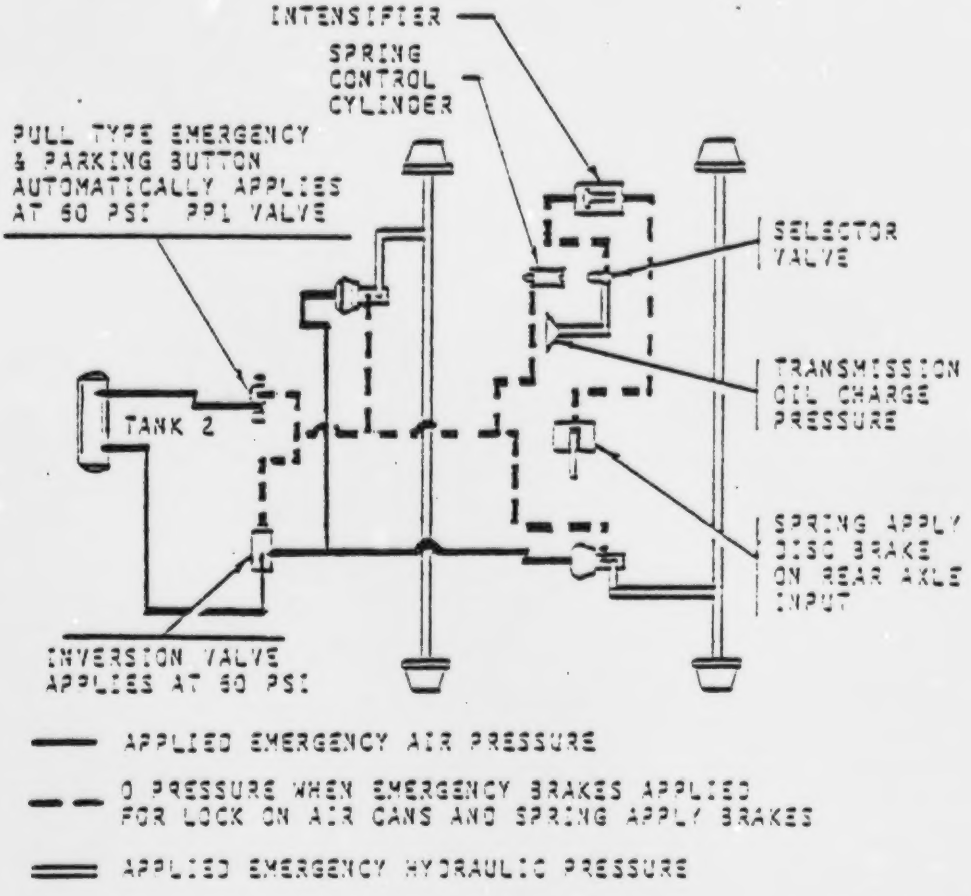

INTENSIFIER
SPRING CONTROL CYLINDER
PULL TYPE EMERGENCY & PARKING BUTTON AUTOMATICALLY APPLIES AT 60 PSI PP1 VALVE
SELECTOR VALVE
TRANSMISSION OIL CHARGE PRESSURE
TANK 2
SPRING APPLY DISC BRAKE ON REAR AXLE INPUT
INVERSION VALVE APPLIES AT 60 PSI
APPLIED EMERGENCY AIR PRESSURE
0 PRESSURE WHEN EMERGENCY BRAKES APPLIED FOR LOCK ON AIR CANS AND SPRING APPLY BRAKES
APPLIED EMERGENCY HYDRAULIC PRESSURE

FIGURE 7

of hydraulic pressure. This "true emergency" brake was introduced in our model JDT-413 truck in 1971 and has become standard in all our equipment. See Figure 7.

This brake consists of a disc mounted directly on either the front or rear axle differential flange. Braking is achieved through a caliper with spring applied hydraulically released pads. Independent of the size of unit, the braking torque of this emergency brake has been designed to give at lease 75% of the braking capacity of the service brakes, (2 axle braking system).

In an automatic or manual application both the emergency and service brakes are actuated.

The question of monitoring these systems was resolved by isolating the brake instrument panel from the rest of the operating instruments, (See Figure 8). In this panel four air gauges are mounted which indicate the air pressure in each tank. Immediately below are four red lights which will come on if the pressure drops below 70 psi. The next row is three green lights which indicate proper stroking of the actuator. Below are three red lights which come on if the actuator overstrokes.

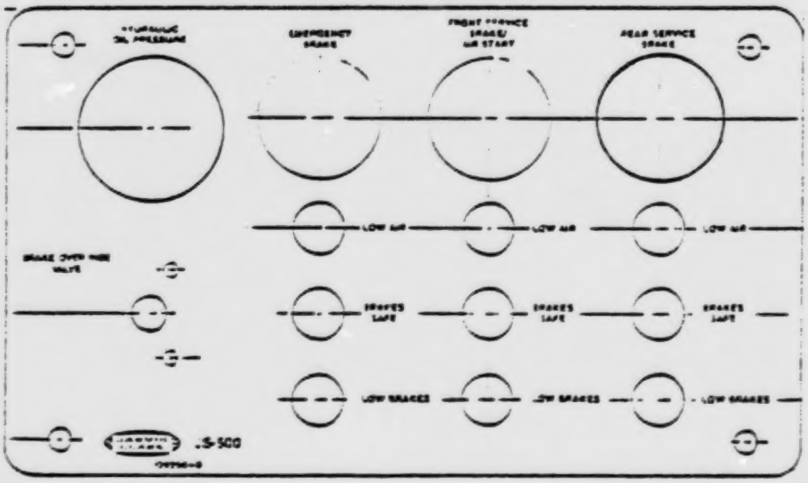

HYDRAULIC OIL PRESSURE
EMERGENCY BRAKE
FRONT SERVICE BRAKE/ AIR START
REAR SERVICE BRAKE
LOW AIR
LOW AIR
LOW AIR
BRAKE OVER RIDE VALVE
BRAKES SAFE
BRAKES SAFE
BRAKES SAFE
LOW BRAKES
LOW BRAKES
LOW BRAKES
.S-500

FIGURE 8

With three separate braking systems incorporated in the design, each capable of stopping the unit, and with proper visual monitoring, we feel we have achieved our goal of providing a truly safe braking system to the mining industry.

USE OF WATER COOLED ENGINES

One of the myths which was prevalent in the sixties and lost some of its credibility in the seventies surrounds the use of water cooled engines in the underground environment.

I believe two factors were responsible for the lack of popularity of these engines in underground application:

1. All the North American manufactured water cooled engines were tested by the U.S. Bureau of Mines at their full horsepower. It was not recognized that when derated the emissions characteristics could be different and improved.

2. The use of a cooling radiator located at the extremity of the unit was deemed to be objectionable because of expected high maintenance requirement.

I believe both these objections had been overcome by the mid 70's and we now see a definite trend towards the water cooled engine in our own manufacturing facilities. At present we supply approximately 50% of our output with air cooled Deutz and the balance is shared by Caterpillar and Detroit Diesel.

I believe the major factor in t[illegible] development of this trend has been the maintenance results and the operating costs experienced in comparison to the accumulated experience over the years with the Deutz engine. We, as a manufacturer, offer these three engines as options in all of our equipment because we feel the industry through its own experience can best develop information through field experience which indicates what componentry is best under operating conditions.

Another factor which may have contributed to the increased population of the water cooled engine is a series of tests sponsored by Texasgulf in 1977. These tests, monitored by the Ontario Research Foundation, showed that the derated water cooled engines and the fully rated Deutz had both noxious gas and particulate emissions which were not that different and also proved that the full rating test carried out by U.S.B.M. did in fact penalize the water cooled engine when ventilation requirements were determined.

We have been using the Caterpillar 3304 series in our JS-220 JarcoScoop since 1974. This engine was selected to try and improve the weight to horsepower ratio in this particular class of unit. If we judge by customer reaction in various markets, this unit has been very popular because of improved productivity and gradeability. Engine life is said to be comparable, or superior, to the air cooled F6L-912W formerly used. As far as the cooling radiator is concerned, there have been very few reported problems which is also supported by sales of radiators to the aftermarket. The radiator currently used with the water cooled engine is the Mesabi type with removable cores, which can easily be repaired if damaged.

OTHER DESIGN IMPROVEMENTS

Stick Steering

Introduced in the early seventies, the stick steering was incorporated in the design of the JarcoScoop for three reasons:-

1. Minimize operator fatigue through the use of a small lever which could articulate the machine from lock to lock through a few inches of movement as opposed to 8 turns of a steering wheel.

2. Eliminate the use of orbitrol type valves which were costly to buy and to maintain.

3. Minimize damage to LHD and components through repeated battering against rock walls which we felt was the result of inability to steer rapidly in emergency situations.

We believe stick steering has achieved all this plus a bonus which was not expected. It has been established that the stick steering units do travel at higher speeds because the operator is more confident of his ability to react quickly to curves or danger of hitting both movable and immovable objects in his path of travel. Higher speed also means higher productivity.

Monostick Boom and Bucket Control

By tradition the first generation LHD used two separate levers for the boom and bucket movement which meant the operator could not steer during the loading cycle because both hands were used to control the boom and bucket functions. This situation was eliminated with the advent of the monostick boom and bucket control, which consists of one lever with four quadrants, one axis controlling bucket movement and the other controlling boom movement.

The net result is that in any part of the cycle the operator is in full control of all the operating functions, i.e. throttle, brakes, steering and boom and bucket movement. Based on report from the mining industry, this relatively simple innovation does result in increased productivity.

Bucket Design

Several years of observation of LHD loading under various conditions led us to conclude that in general load-haul-dump buckets were not designed to fulfil the desireable objective of "true one pass loading". Hence in 1977 we set out to re-design the bucket on the JS-500. One of the basic observations was that the lip lead and side cutter angle of approach had to be changed in order to reduce the resistance to penetration of the muck pile. The net result is the RAM style bucket, which features an approach angle of 45° for the side cutters as opposed to 53° in the original design and an increase in lip lead from 8 to 14 inches. These are now standard on all our LHD models.

One interesting confirmation of our own design was brought to my attention when visiting the Kiruna operation of L.K.A.B. in Sweden in January 1979. After one year of experimentation they had reached exactly the same conclusion as we did on side cutter angle and lip lead.

In conclusion of this paper I would like to mention that all the described design improvements are not entirely of our own making. The Canadian mining industry has contributed a lot in the way of suggestions, testing, reports, etc. which enabled Jarvis Clark to design successful improvements. For this help over the years we are extremely grateful.

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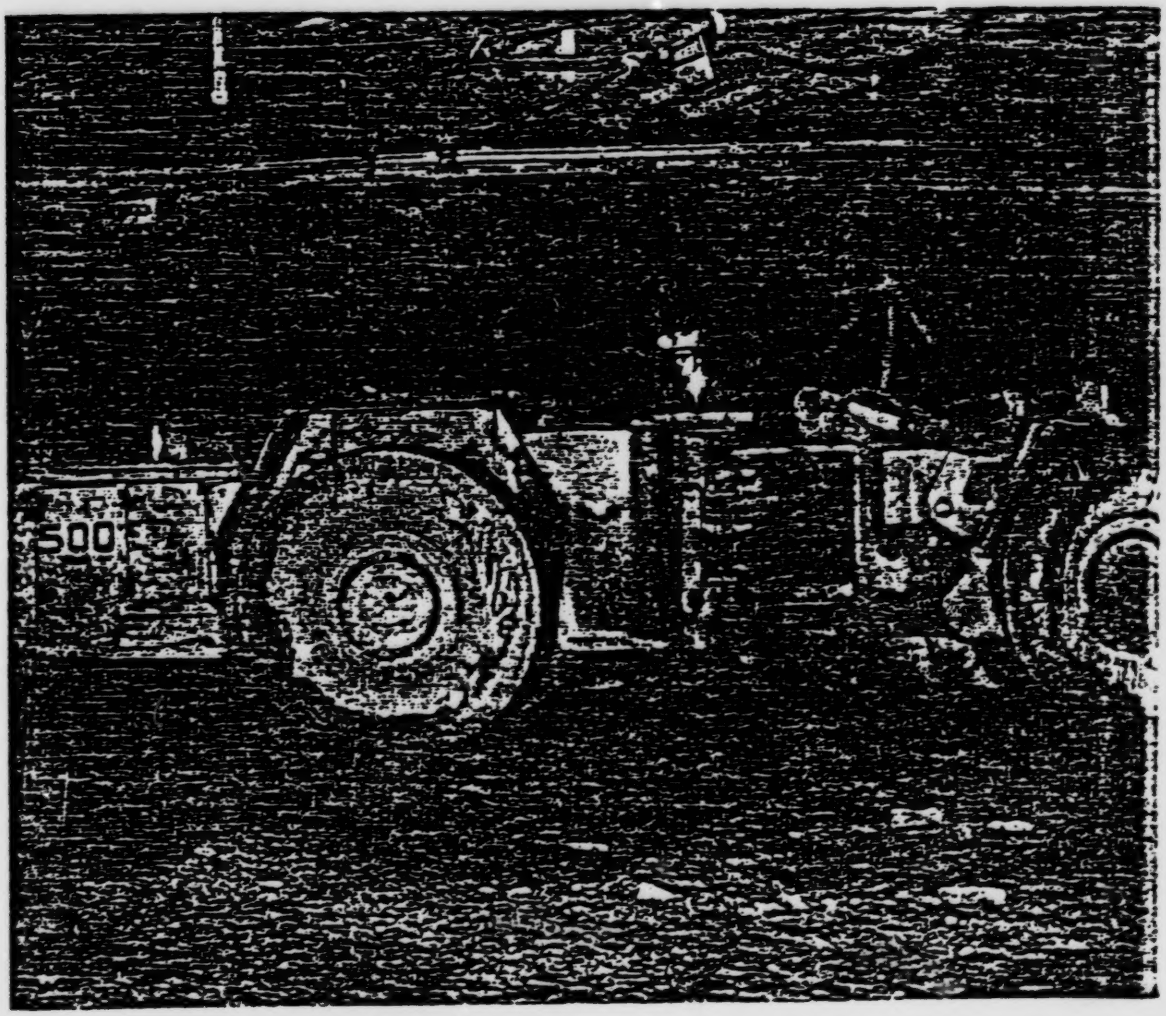
DEVELOPMENT OF ELECTRICAL EQUIPMENT AT SHERRITT GORDON'S FOX MINE

E. N. DuRussel, mine superintendent, and J. L. Muirhead, plant services superintendent, Sherritt Gordon Mines Ltd.



Increased ventilation requirements and rising air-heating costs have spurred Sherritt Gordon Mines Ltd. to introduce electric-powered haulage equipment at its underground Fox mine in northern Manitoba. This story highlights the development of a three-phase, four-conductor, ac trolley system for haulage equipment at the Fox mine.

The mine site is 30 mi southwest of Lynn Lake, Manitoba. It has been operated by Sherritt Gordon Mines Ltd. since 1970, producing copper-zinc ore at the rate of approximately 1 million tons/yr.

The orebody occurs as a large, east-west, lenticular zone of semi-massive sulphides, approximately 1,500 ft long near the surface and 15-100 ft wide. Dipping 70° to the north, the orebody appears to bottom out at about 3,200 ft below the surface. It is covered by 15 ft of overburden.

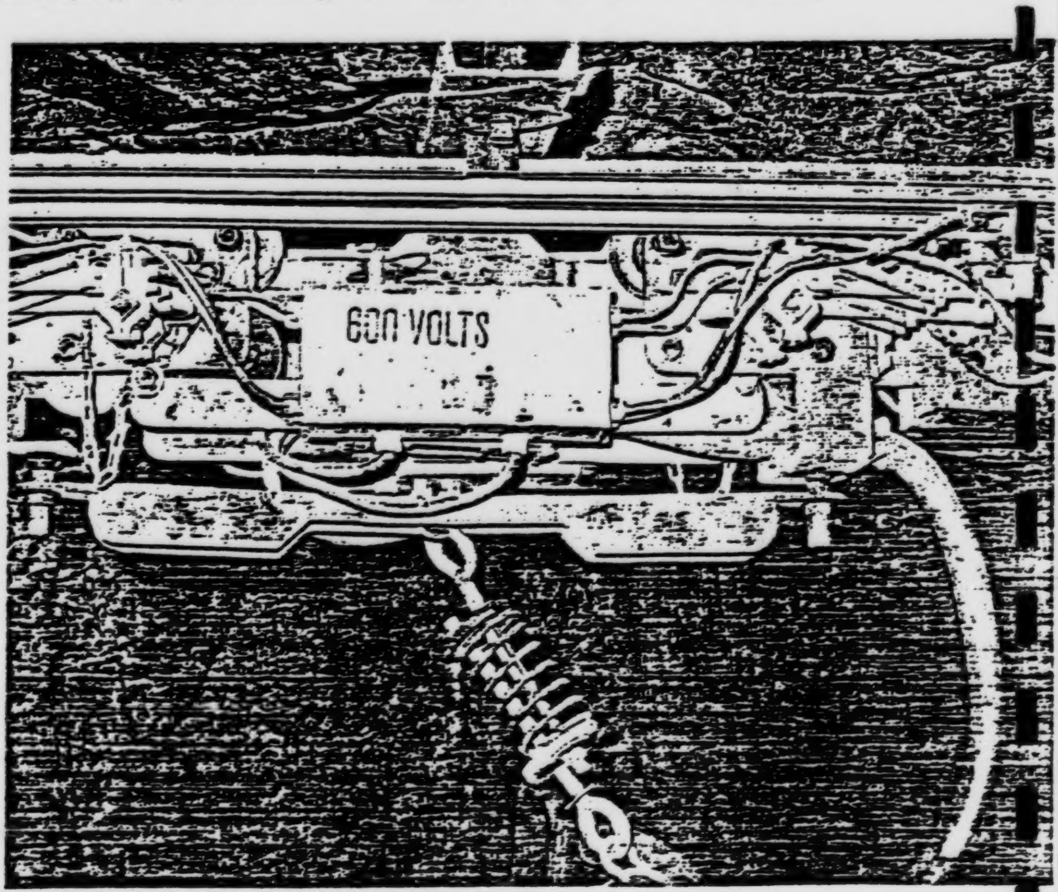
Open stoping with delayed hydraulic backfilling has been the mining method. Both 2-in.- and 6½-in.-dia drilling are used. Underground ore transport has been by diesel-powered 5-yd^3 LHDs and 26-ton trucks. In 1979 and 1980, two electric-powered, 5-yd^3 LHDs with 500-ft trailing cables and take-up reels were purchased and utilized at the 400 level.

The Fox ore occurs as a disseminated-to-massive sulphide body, primarily consisting of pyrite, pyrrhotite, chalcopyrite, and sphalerite, with minor amounts of arsenopyrite and galena. The host rock consists of metasediments with minor amounts of metavolcanics intruded by post-ore dioritic-gabbroic dykes.

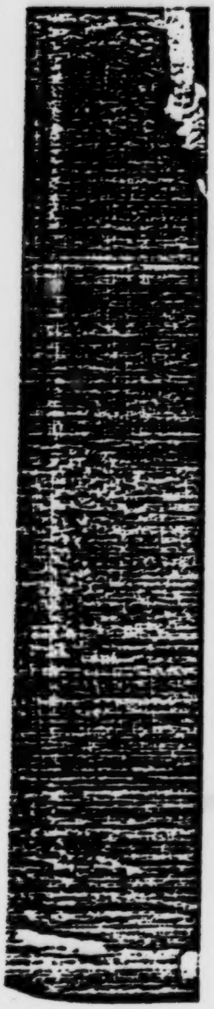
The orebody is enveloped by an alteration zone consisting of quartz, sericite, albite, muscovite, biotite, and amphibole. Basic dykes occur within the ore and alteration zone, and outside of the alteration zone. The ground is generally competent and well suited for blasthole mining.

ACKNOWLEDGEMENT

This work was presented by E.N. DuRussel and J.L. Muirhead at the Fifth Annual Underground Operators Conference of CIM in Sudbury, Ontario (Feb. 16-18, 1981) and appeared in the May 1981 issue of the CIM REPORTER. E&MJ thanks CIM for permission to publish this work in its entirety.



A captive trolley pick-up mechanism is pulled along the monorail by the power cable to the vehicle.



This electric trolley LHD is powered by trailing cables.

FOX DEVELOPMENT AND MINING

The original development at Fox mine included a vertical 13 x 18-ft. five-compartment shaft for hoisting men, materials, ore, waste, and intake ventilation to 2,343 ft below the collar. Above the 2,000 level, main drawpoint levels were developed at 400-ft intervals, with three sublevels between the main levels positioned for 2-in.-dia blasthole mining. The ore mined from these upper stopes is trammed by diesel-powered, 5-yd^3 LHDs to a central orepass system.

To ease the ventilation demand, two electric Jarvis Clark JS500-E LHDs powered by electric cables on take-up reels were installed on the 400 level in 1980 and have been dumping into the same orepass system.

The orepass feeds a 42 x 48-in. jaw crusher on the 2,100 level. The crushed ore is conveyed to the shaft and skipped to the surface from the 2,200 level.

Exploration and feasibility studies proved minable ore below the 2,000 level and led to the development of a 15% decline system to mine the ore in 200-ft-high blocks. A paper entitled "Ore haulage by trucks at the Fox mine," by P.H. Goodwin, presented at the 1977 CIM Underground Operator's Conference, outlined these feasibility studies and the initial truck haulage from the 2,200 level. As described in the paper, all ore below the 2,000 level is hauled up the decline and fed to the orepass system at the 2,000 level.

Currently, truck haulage from the 2,400 level is scheduled for 1,100 tons/day, utilizing two trucks on three shifts, with one truck for spare. To meet this production target, good fragmentation of ore in the stopes is required.

An unpublished paper entitled "Hoisting deep ore at Fox mine," by John G. Kelley, Planning Engineer, discussed the need for an increased production rate in 1982 from the 2,600 level and described the avenues that Fox mine was investigating to allow economic mining of the lower part of the orebody. (The paper was presented at the District Four Annual Meeting of the CIM in Flin Flon, September, 1980.) The possibilities presented were: 1) to deepen the main shaft skip compartments; 2) to expand the present diesel trucking system; 3) to convert to electric-powered trucks; or 4) to introduce winze hoisting through an 8-ft-dia borehole. The development of a practical three-phase, four-conductor, ac trolley system for haulage equipment was detailed in the remainder of the paper.

SELECTING ELECTRIC EQUIPMENT

There are several reasons why the Fox mine opted for the introduction of electric-powered haulage equipment:

- To mine the deeper ore with diesel equipment on an extension of the present decline system would have required extensive alteration of the ventilation system. The cost for raises, fans, heaters, and control bulkheads was estimated at $1.5-2.0 million.
- The extra ventilated air required for diesels would cost an estimated $200,000/yr to heat.
- The Fox mine has a limited life of only 5 yr, and Sherritt has been made aware (from closing its Farley mine operations) that used diesel equipment and ventilation raises are worthless. Electric equipment, however, could have resale value.
- The Fox mine has an attractive long-term hydro power contract.
- The use of more diesel equipment is not environmentally attractive to the mining work force.
- Canada has not yet received the full impact of increased petroleum costs that have been experienced around the world, nor has it experienced any shortages.

For a number of years, Sherritt had been interested in the development of an electric-powered truck for the Fox decline, but the idea was discouraged by potential suppliers because of its limited market potential. Then, an article in a trade magazine revealed that a cement plant at Riverside, Calif., had used an electric trolley truck to haul limestone from an underground mine. The operation started in 1955 and continued for 18 yr using a two-wire, dc system with twin trolley poles. Due to changes in the mining plan, the plant eventually reverted to a conveyor haulage system.

The 10% decline of the Riverside operation had been wide enough to permit two-way traffic. The four trucks on the system were able to follow one trolley line down the decline and another one up it, with only a few operational problems. A visit to the plant confirmed Sherritt's belief that electric trolley trucks have a place in underground mining and that their development should be catalyzed.

Early in 1979, Fox mine representatives visited the Patino mine in Chibougamau to view the operation of a 5-yd^3 electric LHD (JS500-E). The visit confirmed that this vehicle was a practical replacement for the diesel LHDs that were in operation at the Fox mine. Sometime after this visit, Sherritt and Jarvis Clark agreed to give serious thought to the development of an electric trolley truck.

A two-wire, dc system was considered at the first meeting of Sherritt, Jarvis Clark, and Canadian General Electric. It was later decided that this system would not be able to give an adequate drive performance. A three-phase rectifier drive was believed to out-perform and be safer than the two-wire, dc system. Only four conductors would be required for a high-frequency ground checking system, where the signal is induced in the power conductors and monitored by a single ground wire.

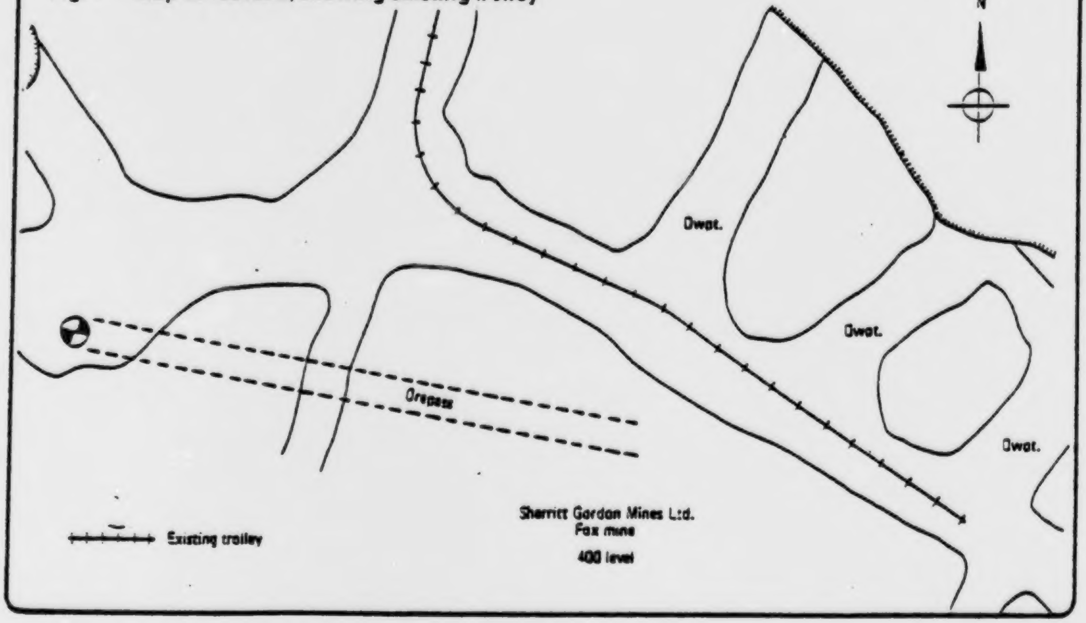
The development of a practical three-phase, ac trolley pickup system with four conductors was agreed upon. Three of these would be used for the power, and one for the ground. Ground protection would be by a frequency injection method.

Some of the more important visits included: 1) The power unit manufacturer (CGE), to agree on the acceleration/retardation characteristics of the truck, and the degree and type of retardation required; 2) the University of West Virginia, to view their "rapid transit" system, which uses rubber-tired vehicles driven by ac power through conductors mounted on an "I-beam" to one side of the vehicles; 3) a test facility at Pittsburgh, to see "people movers" under test. Again, these vehicles were driven by ac electric power but from "between-track" conductors. These vehicles are being used at the Seattle, Miami, and Atlanta airports; 4) a crane and monorail manufacturer near Cleveland, Ohio, to see the manufacture and use of underslung bridge cranes and monorail transportation systems. The cranes and monorails used fixed conductors, and the running surfaces of the monorails were a specially formed and hardened T-section steel. This manufacturer has already developed a monorail type of trolley pickup and monorail switching systems.

These visits showed that the technology was available for developing a practical ac trolley system for underground use. Switching devices would allow vehicles to meet and pass on the same trolley system, and a 100-ft cable spool on the truck would facilitate the passing of trucks, and their loading and dumping.

Testing the trolley. Sherritt and Jarvis Clark were convinced that an electric trolley truck was practical. Since it would take 60 weeks to obtain the motor and control systems, and because a considerable amount of capital expenditure in the project was involved, it was considered prudent to test the trolley system prior to completion of the truck manufacture. By March 1980, Fox mine had purchased and was success-

Fig. 1 — Map of 400 level, showing existing trolley



fully operating two 5-yd³ electric LHDs powered by trailing cables. This installation was believed to be ideal for trolley use, and the decision was made to convert one electric LHD to trolley operation for drawpoint mucking, hauling, and dumping.

An order was placed for 200 ft of monorail track, including four conductors and a prototype trolley pickup. The track and pickup designs were based on standard monorail components and conductor bars modified to suit the circumstances. The track was installed without switches but included a 90° turn and stopped short of the dump (see Fig. 1).

Tests were designed to: 1) Evaluate the operation of the trolley, monorail, and conductors for operational practicability in an underground environment; 2) test the combined operation of the take-up cable reel and trolley to allow the LHD or truck to operate away from the monorail, but within a fixed radius (Sherritt felt it was not practical to have the trolley rail pickup in the loading or dump areas of the truck haulage, nor should the trolley installation go into a drawpoint. Also, a very flexible system was required to operate a LHD on a trolley); 3) obtain operator and maintenance feedback before committing the majority of the capital.

One of the concerns has been to establish operating procedures for mechanics, electricians, and operators with regard to daily checks, scaling loose, blasting, etc. These procedures are being set up with multi-vehicle operation in mind, as well as the obvious objectives of being acceptable to the work force and achieving the desired results.

The live trolley system introduced a new hazard to the workplace. Therefore, many signs have been posted, the lighting has been improved, and flashing red lights have been installed to indicate when the conductors are powered.

Test results. From the start, the cooperation of installation crews, operators, mechanics, and electricians was excellent. All involved realized that we had to start slowly, and take it one step at a time.

Minor failures and problems occurred, but the machine operated successfully from the start. The operators had initial misgivings about the overhead conductors, but they

Fig. 2 — Cross section of trolley pick-up rail

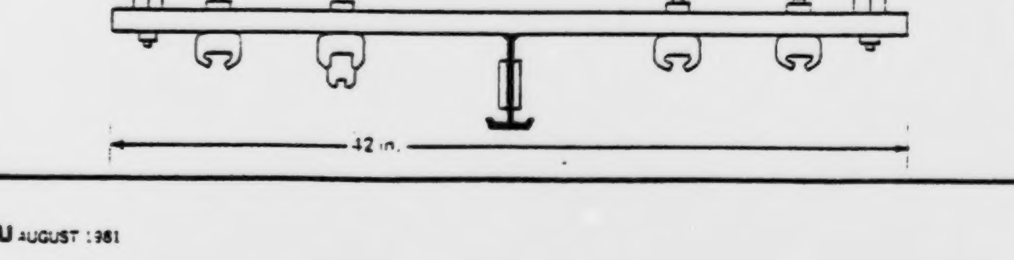
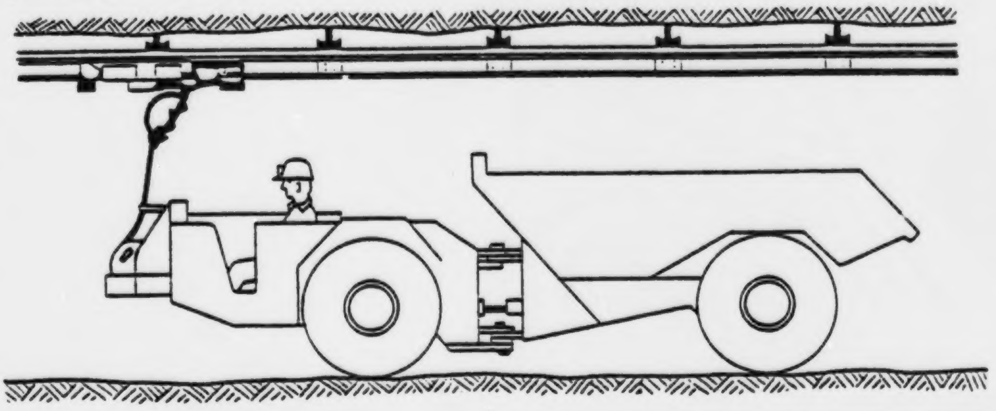


Fig. 3 — Sketch of proposed trolley truck

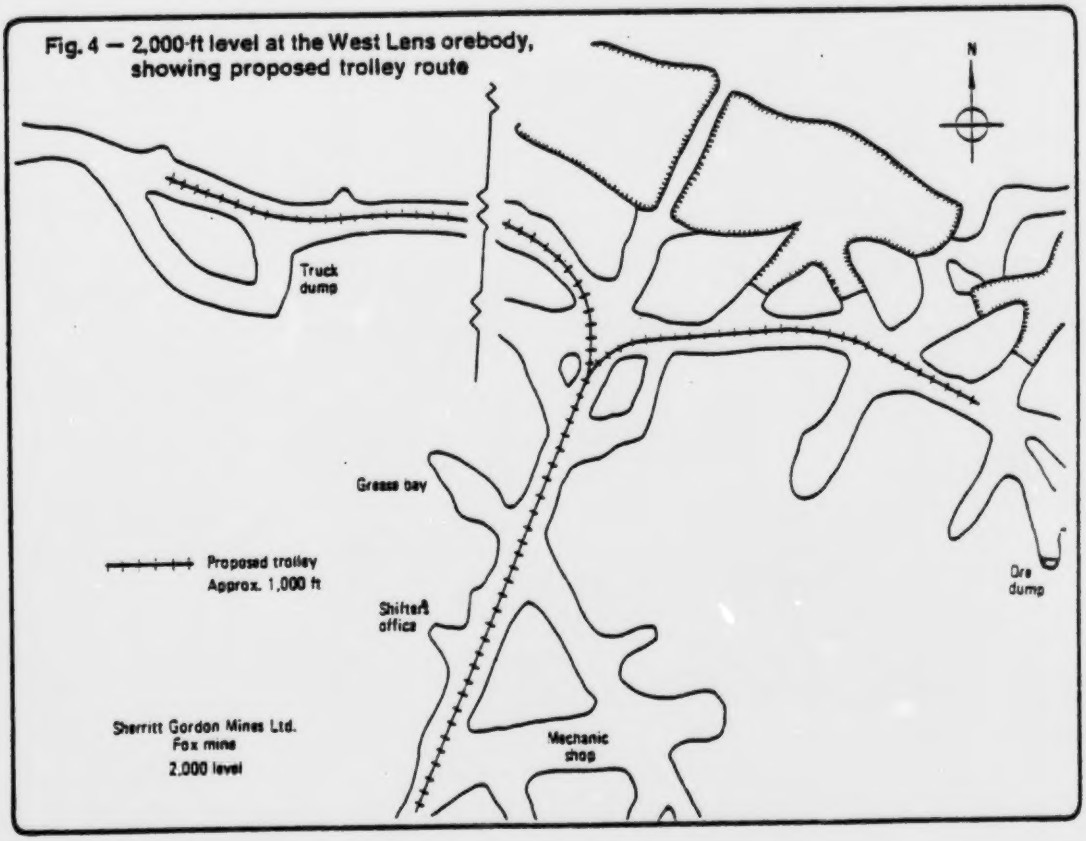


soon realized that there was little danger, unless someone would deliberately try to reach them. It is possible that problems could be encountered over the long range from the continual flexing of the cable near the trolley.

Following the initial test, capital approval was given for ordering the prototype truck, and more track was ordered to extend operation of the trolley-powered LHD to the full length of the level. The test proved the viability of the trolley-powered truck, and it demonstrated the practicality of the trolley-powered LHD.

The monorail. An inverted "T" steel section, called the "patent-rail," is the running rail for the trolley cart. Pieces of 6-in. steel channel are rockbolted to the back of the drift at

Fig. 4 — 2,000-ft level at the West Lens orebody, showing proposed trolley route


N
Truck dump
Grease bay
Proposed trolley Approx. 1,000 ft
Shifter's office
Ore dump
Sherritt Gordon Mines Ltd.
Fox mine
2,000 level
Mechanic shop

approximately 6-ft intervals, and the patent rail is bracketed about 6 in. below these channels. Spacer pipes are used on the rockbolts to keep the channels horizontal and firmly fixed. In areas where the track is curved or where the LHD moves away from the track and pulls sideways at it, the support interval is reduced. The conductors are also supported from the cross-channels by insulating brackets. The conductors (Saf-T-Bar type from Howell Corp.) are insulated on all but the lower face and are held in a fixed position relative to the patent rail. Two of the conductors are suspended on each side of the rail (see Fig. 2).

The trolley. The captive trolley pickup is pulled along the monorail by the power cable to the vehicle. Four double sets of spring-loaded pickup brushes are mounted on the trolley to draw the power from the conductors. Vertical and horizontal wheels in the trolley roll on the T-rail and allow little lateral movement relative to the T-bar. Since the spatial relationship of the running rail and conductors is constant and the trolley is trapped, there is little likelihood of losing contact.

The ground test system. Sherritt's present electric LHDs operate using a five-core cable—three for power, one for ground, and one for ground monitor. But, in order to simplify the trolley system and take advantage of present-day technology, it was decided to use only four conductors—three for power, and one for both ground and ground monitoring. Because of the common ground and ground monitor conductor, a specialized ground monitoring system is being used.

The high-frequency signal is transmitted to the vehicle through the power conductors. If the vehicle is properly grounded, the return of the signal is monitored through the ground conductor. If no reply is received, then the power to the monorail conductors is interrupted. This type of system is being successfully used on cable-fed electric Scooptrams in Ontario as well as at this installation.

Although not currently available, it is thought that the monitoring units can be built with various frequency signals. This will allow the independent use of several vehicles on the same monorail system at the same time. If one of these vehicles fail to reply to its ground monitor signal, the power will be interrupted for all vehicles on that system. After disconnecting the faulty vehicle from the system and switching off its ground monitoring unit, normal operation can be resumed with the remaining vehicles.

Proposed test truck operation on 2,000 level. The prototype truck is based on the standard Jarvis Clark JDT-426, a 26-ton-capacity, diesel-powered truck, of which Fox currently has three (Fig. 3). It will be modified by replacement of the Deutz diesel engine with a CGE 270-hp, dc electric motor. Other components of the drive train, such as the torque converter and transmission, will not be changed.

All the standard braking systems will be featured with the diesel truck (i.e., independent front axle, rear axle, and drive line brakes). An essential part of the diesel truck operation is the use of the engine as the retarder. The electric system has also been designed to give a similar retarding effect.

On the prototype vehicle, both regenerative and dynamic braking could be available. Control of motor speed will be from a standard foot pedal. The vehicle will receive its power from the trolley system via a 100-ft-long cable, which will be normally wound on a single-layer horizontal spool across the front of the machine. The ac power will be converted to dc by a standard solid-state control package, as used on the JS500-E LHDs.

In use, the truck will be loaded at the West Lens orebody on the 2,000 level and will dump in the orepass after traveling 1,200 ft (Fig. 4). Test instrumentation mounted on the vehicle will indicate what modifications will be needed to adapt to decline haulage.

In summary, the developments over the last two years have been of great interest, and Sherritt is now confident that one or more trolley-powered units can operate on a single-track installation, using different frequencies to monitor the different vehicles. By switching the rail and conductors, the vehicles will be able to pass each other at designated areas.

The authors would like to thank the CIM Underground Operators Committee, Jarvis Clark, and Sherritt for allowing them the opportunity to present this work at the Fifth Annual Underground Operators Conference of the CIM. ■

THE FOX MINE STORY

The copper-zinc orebody at Fox was discovered by Sherritt exploration personnel in 1961, using geophysical methods that included airborne and ground surveys. Surface diamond drilling indicated ore reserves between surface and the 2,000 level at 12,269,000 tons, grading 1.74% Cu and 2.35% Zn. Subsequent diamond drilling from 2,100 exploration level outlined an additional reserve of 3,084,000 tons at 2.262% Cu and 1.117% Zn. Ore reserves as of December 31, 1978 totaled 6,333,000 tons at 1.79% Cu and 2.22% Zn.

The Fox orebody—a Wasekwan volcanogenic proximal exhalative deposit—occurs in a belt of isoclinally folded metavolcanic and metasedimentary rocks. It strikes N72°E and dips 70°NW. The host rocks are considered Aphebian in age, having undergone regional Hudsonian deformation and metamorphism.

The sulphide zone is about 100 ft wide and 1,500 ft long. It extends 3,000 ft down dip. In the lower levels, the orebody splits into two lenticular orebodies—the Main lens and the West lens—which are conformable with the enclosing rock units and gneissosity. About 70% of the ore zone is massive sulphides. The remaining 30% is disseminated sulphides.

Massive sulphide ore consists of pyrite with varying amounts of pyrrhotite, chalcopyrite, sphalerite, arsenopyrite, and quartz-feldspar gangue. Disseminated sulphide ore consists of some of the sulphides present in the massive ore, but it contains a higher proportion of silicate.

An alteration zone envelops the ore zone. It is approximately 250 ft wide and 1,300 ft long, and it extends well below the 2,900 level.

Higher copper values occur in the core, in the eastern extremity, and in the western mid-section of the ore zone. Zinc values are found along the footwall and hanging wall contacts and concentrated on the western side of the copper zone. Copper grades increase and zinc grades decrease with depth.

This is the Fox mine story:

BRITISH TUNNELLING SOCIETY MEETING

Construction of the Megget water tunnel

The Megget Scheme has been planned on behalf of the Lothian Region by their consultants, R H Cuthbertson of Edinburgh, and will meet the requirements of that city and its environs well into the next century. The location of the project, the layout of the works and existing water supply schemes are shown in Fig 1 (see *Tunnels & Tunnelling*, December 1978, p14.)

The discussion was opened by Mr Young who, after describing the attractions of the area in which the works are situated, the excellent facilities for visiting engineers such as the Tibbie Inn, went on to describe briefly the dam with its asphaltic core and elegant combined outlet and overflow tower. He then described in more detail the tunnel section of the aqueduct which will carry the water north from the reservoir for a distance of over 47km towards Edinburgh.

An investigation into the rock conditions was carried out under the direction of Professor Knill of Imperial College, London. A total of five boreholes was drilled in the main rock types anticipated. Investigations into the quartz contents were also carried out and comment was made regarding the 5 per cent figure obtained from a normal optical test in comparison with the 41 per cent quartz content obtained by X-ray refraction.

All the available information on the anticipated geological conditions, including the various boreholes, logs and other tests carried out, were given to the tendering contractors who were then responsible for making their own assessment as to the required temporary support. However, a rock classification system was devised by Professor Knill to take account of possible changes in the rock types and this was described briefly at the meeting of the Society on 21st February 1980 (*Tunnels & Tunnelling* July 1980). Although the proportions of the various rocks encountered had turned out rather differently from that anticipated, in overall terms the rock classification system had been successful: the final cost of excavation only varied between 1 and 2 per cent from the figure in the bill of quantities.

The contractor had made excellent progress with the entire project which had been completed ahead of schedule at a final cost, ignoring Baxter increases, of less than the tender value.

Planning

Mr Lawrenson, before going on to illustrate the project with a few slides, commented on the extremely harmonious working relationship reminiscent of times gone by which had existed throughout the entire three and half years whilst the tunnel was being built.

A meeting of the British Tunnelling Society was held on the 15th January 1981 to discuss the design and construction of the tunnel works on the Megget Scheme. The Chairman was Douglas Parkes and the subject was introduced by Alex Young of R H Cuthbertson and Denis Lawrenson of Edmund Nuttall.

The contract was awarded to Nuttalls in December 1977 at a cost of £6.7 million and consisted of the driving of 7.7km of tunnel with a finished diameter of 2.4m at a gradient of 1 in 642 rising from the Manor End. The depth of the tunnel varies between 80 and 100m below ground level and the tunnel passes under a number of the main tributaries of both the Manor and the Megget waters.

The contract provided a rare opportunity to carry out almost a repeat performance of the Wyresdale Tunnel Contract which was very similar although slightly shorter, although of course by 'sods law' the Megget Tunnel had a lining of 2.4m diameter whereas Wyresdale was 2.6m diameter, thus preventing the use of the same tunnel shutter! Mr Lawrenson said that as Wyresdale had been a very successful contract at least progresswise they had no hesitation in deciding to use the identical concreting methods for Megget and the same muck-handling equipment, namely Hagglunds gear, and a similar method of tunnel ventilation, but the winning of the muck merited fresh consideration.

The possibility of using full face tunnelling machines was considered but was rejected as it was felt that the strength of the sandstone, up to 37 000psi was too strong and there was also worry about the possible 'mincing machine effect' when driving machines through the fault zones and lengths of black shale. Progress on the Wyresdale Tunnel had averaged 58m per week on each phase with a best week of 90m using four number Atlas Copco BBD 90W drills with 2.1m diameter steels mounted on air legs, but the rock there was softer and predominantly mudstone.

Following extensive investigation it was eventually decided to use Atlas Copco rail mounted twin boom drilling jumbos each fitted with two COP 1038 HY Hydraulic Rotary Precussion drills (Fig 2). The RH 315 rotary booms were designed for 360° rotation and were equipped with mechanical linkage for automatic parallel holding of the feeds and also an effective anti-rod jamming device. The adoption of this equipment allowed the gang size to be reduced considerably from a shift boss, leading miner and four miners on Wyresdale to a shift boss and two miners on Megget. There was also the advantage that the working conditions at the face would be greatly improved as the noise level would be considerably lower (10-15db) although in practice the men had to wear ear defenders.

The use of hydraulic drills would also eliminate the oil and water fog associated with pneumatic drilling machines, consequently making the atmosphere a lot more pleasant to work in and also increasing the accuracy of drilling. Before driving commenced a training programme was set up by Atlas Copco and the necessity of adequate maintenance for this sophisticated type of equipment was emphasised.

Tunnel driving

The contract period commenced in late February 1979 and driving was carried out from both ends on a 15 x 8 hour shift per week basis, the objective being to complete a cycle time for drilling and firing and mucking of less than four hours. The tunnel section was 3.3m x 3m 'D' shaped as it was necessary to provide a reasonable clearance between the Hagglund cars and the various services at the entrance to the tunnel.

The drilling pattern was generally 40 x 45mm diameter holes 2.6m deep with three 75mm uncharged relief holes at the centre. Power for the rigs was taken into the tunnel at 3.3kV and stepped down to 415V through a transformer which was generally kept within 500m of face.

The explosive used was Quarrex A, which was delivered to the site in tubes of rigid polythene and came in two standard lengths, the normal being 600mm long and the other 500mm complete with detonator pocket. These tubes could quickly be made into lengths of 1700mm for each hole and about 70kg was used for each full round.

Tunnel ventilation was generally based on Kortmann Fans set in pairs approximately 1700m apart. These operated an extraction system using 600mm diameter flexible ducting and this was supplemented by a 600mm diameter Woode Fan attached to ducting at the face to assist with the clearing of the fumes.

The electrically operated Haggloader which had also been used at Wyresdale gathers the muck with its arms and transfers it via a single chain AFC into the Hagglund shuttletrain. In the bottom of the Hagglund car there is a compressed air driven chain AFC, which moves the rock and transfers it from one car to the next, the discharge end of each car fitting into the loading end of the car behind to make the conveyor in effect a single unit which runs the length of the train. This arrangement eliminates the need to move the train in and out of the tunnel to change full and empty cars. At Megget four cars were used and were pulled by a single SIG 15 tonne battery locomotive.

The hydraulic drilling was very successful, allowing a peak production rate of 105 lineal metres on one face in a five day working week and breakthrough occurred in October 1979, 18 weeks ahead of programme. In the last two months of the operation the drill, blast and muck cycle time was lowered to approximately three hours.

Mr Lawrenson went on to describe the waterproofing measures in the wet lengths of the tunnel. These consisted of stitching thin polythene sheeting to the crown and side of the tunnel with steel mesh and occassionaly small bore plastic tubes to form the pathway for the water to gravitate to the invert. However, as it turned out the waterflow was not a serious problem and the combined flow from both drives never exceeded four million gallons per week.

Concreting

Prior to the start of the main concreting operation the tunnel was cleaned out using the Haggloader with its arms as scrapers and loading again into the Hagglund cars. Road forms were then laid to line and level using the same lasers which had kept the drive in the correct direction. This slab provided a firm base to relay the raid truck for the concreting plant and also to provide an accurate surface on which to set the shutters. The excellent progress achieved was possible because the Haggloader was the ideal tool for the job and because the engineer adopted a sensible and practical approach in his requirement for invert preparation and did not insist on removing good compacted rock dust below the payline just to replace it with good compacted concrete.

The main concreting operation started from the Manor Portal in January 1980 and used a 60m long folding shutter designed and built at Nuttalls' Kilsyth Plant Depot (Fig 3). Concrete was batched at Megget Portal using a Benford PB 42 batching plant and loaded directly into delivery trains made up of two Mulhauser BM5/1000 transit cars. The concrete which had a specified strengh of $25N/mm^2$ was placed through a 100mm diameter pipe directly over the top of the shutter using a Pressweld pneumatic placer. Concreting finished on 12th August 1980, 35 weeks ahead of programme, and averaged 252 lineal metres per week with a best week of 360 lineal metres.

Tunnel grouting

Holes for cavity grouting were generally drilled at 3m centres in the crown of the tunnel although test holes were regularly drilled in the knee position to check for any voids; in general few were found. Contact grouting holes were drilled at 6m centres along the crown of the tunnel but as expected grout takes were very low. Fissure grouting was carried out in areas where fissures were identified from the geological records and where water appeared to be entering the tunnel under pressure. Fissure grouting was also carried out at the junction of the steel and concrete linings to form a grout curtain near the portal where the ground cover was relatively low.

Mr Lawrenson concluded by saying that Megget was in three words 'a happy

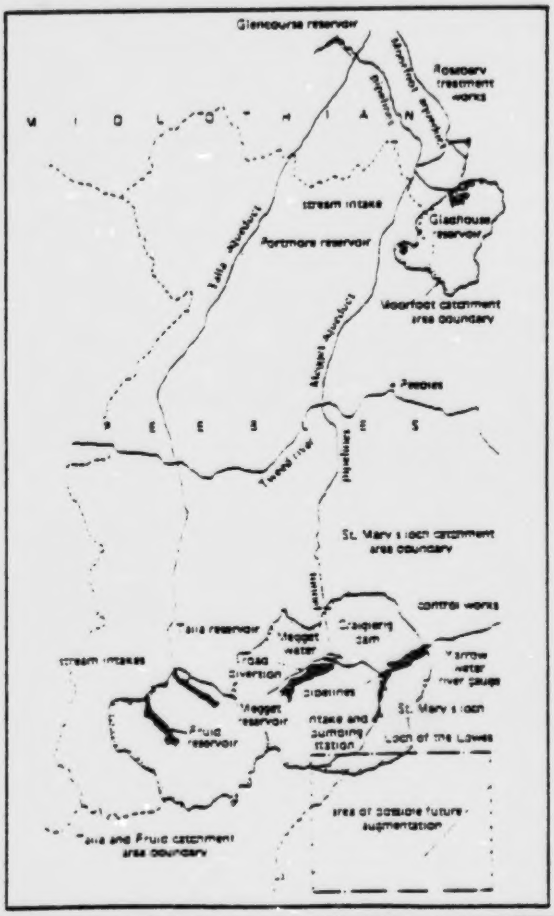
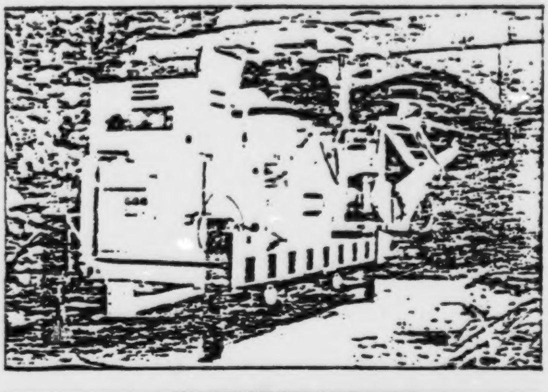
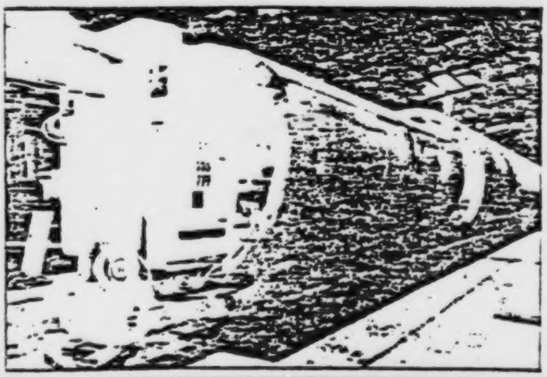

Glencourse reservoir
Moorfoot aqueduct
pipelines
Rosebery treatment works
MIDLOTHIAN
stream intake
Gladhouse reservoir
Portmore reservoir
Talla Aqueduct
Moorfoot catchment area boundary
Megget Aqueduct
Peebles
PEEBLES
Tweed river
pipelines
St. Mary's loch catchment area boundary
pipelines
control works
Talla reservoir
Megget water
Craigierig dam
Yarrow water river gauge
stream intakes
Road diversion
pipelines
Megget reservoir
Fruid reservoir
intake and pumping station
St. Mary's loch
Loch of the Lowes
area of possible future augmentation
Talla and Fruid catchment area boundary

Fig 1 (left). The Megget Tunnel in relation to existing water supply schemes. Reproduced from Water Power & Dam Construction, *September 1978, by kind permission of the Editor.*

Fig 2 (below). Atlas Copco twin boom hydraulic drilling jumbo.

Fig 3 (bottom). The tunnel shutter at Nuttall's Kilsyth plant.





job'. It was a job run by experienced people on both sides and not one minute was lost in a dispute of any kind. The finish of the concrete lining was of a very high order and all concerned at the site could be proud of an excellent contract completed almost nine months ahead of the original programme.

Paradoxically these successful operations were to result in a substantial loss on the concreting operations owing to the way in which the Baxter formula works, in that it failed to take account in the short time period of a staggering rise in the price of cement which took place immediately after the start of lining. Mr Lawrenson paid tribute to his agent Mr Eric Hull and also the resident engineer, Peter MacPherson and the spirit of co-operation that existed throughout.

Discussion

The discussion was opened by Mr Graham West of TRRL who showed a slide comparing the Wyresdale and the Megget Tunnels. He asked Mr Lawrenson if the difference in progress between the two jobs was mainly the result of the use of hydraulic rock drills. Mr Lawrenson replied that this was so although Mr Moore of Nuttalls added that only 4 per cent of the Megget Tunnel had had ribs, whereas ribs had been used extensively at Wyresdale.

Mr Jones of Atlas Copco commented that the advance rate at Megget had been limited by the fact that the four Hagglund cars could only deal with 2.7m pull. With penetration rates 2m/mn there was scope for significantly faster driving.

Dr Bill Ward asked how dealing with water in the tunnel had been paid for and also commented that the rock classification appeared to be based on the frequency of discontinuity. Mr Lawrenson said that dealing with water had been a bit of a gamble and that they had allowed for a 6in pumping main in addition to gravity and that this had been adequate. Mr Young added that over the years they had evolved a practice of giving the tendering contractors a document including all known information on the project and that this had included information from the testing programme. Information on water flows had generally been as forecast.

In response to a question from Mr Collins it was emphasised by Mr Young that the support was entirely the contractor's responsibility and that there was no measurement of support installed in the tunnel. However, tendering contractors had been asked for details of the temporary support which they anticipated installing with each rock class and this had been discussed with the contractor during the tendering period.

Also in response to a question from Mr Collins Mr Lawrenson said that no stop ends had been used at weekends and the concrete had been left at its natural slope but the surface was air and water jetted prior to the restarting of concreting. Some XPM was used in the invert.

Dr Ashburner of Atlas Copco commented on the figures for quartz content and asked whether Mr Young would now insist on an X-ray test. Mr Young replied that Professor Knill had supervised the testing programme but that they knew of the small particle size for the quartz and that this was the reason why it was picked up on the X-ray and not on the optical system.

Fairly good profile

In response to a question from Barry New of TRRL Mr Lawrenson said that the boom driller had allowed them to obtain a fairly good profile although he had no information available on overbreak. Half second delays were used throughout and the charge per round was about 70kg. In response to another question Mr Lawrenson said that of course explosives were kept very strictly under control but that no particular problems had arisen!

The Chairman, Mr Douglas Parkes, thanked both Mr Young and Mr Lawrenson for their excellent presentation and the contributors from the floor for a very lively and interesting discussion. ☐

N S M Berry, BTS Member.

BRITISH TUNNELLING SOCIETY Notes

1 Change of date for informal discussion

Members are asked to note that the informal discussion 'Labour in Tunnelling in the Eighties', to be introduced by Mr G R Henderson, National Secretary of the Transport and General Workers Union, will now take place on Thursday, 21 May 1981 in order to avoid a clash of dates with the ITA meetings in Nice.

2 Programme of meetings

a) Thursday, 23 April 1981
Harding Prize Competition: presentation of finalists' papers.

b) Thursday, 21 May 1981
'Labour in Tunnelling in the Eighties': informal discussion to be introduced by Mr G R Henderson, National Secretary of the Transport and General Workers Union.

c) Thursday, 17 September
The Caracas Metro System: informal discussion to be introduced by Mr A J Howard.

d) Thursday, 22 October 1981
Annual General Meeting followed by an informal discussion of Falconbrook.

e) Thursday, 19 November 1981
Informal discussion of the Tyne and Wear Metro. These last two subjects have been switched around as the Institution of Civil Engineers is holding a meeting on Tyne/Wear on 10 November, which all BTS Members are welcome to attend.

d) Thursday, 10 December 1981
Film Evening.

3 Harding Prize Competition

The papers selected for presentation at the Competition on Thursday, 23 April will be chosen from the following:

'Tunnelling through frozen ground' a paper on a project for the North West Water Authority by C A Garnett of Mott, Hay and Anderson,

'Concrete lining of the Kielder Tunnels' by G G T Masterson of Baptie, Shaw and Morton,

'Partial relining of Falkirk High Tunnel' by K R Scott of James Williamson and Partners, and

'The mechanics of tunnelling in silt' by R N Taylor of the University of Cambridge.

4 Views from the Site

The Society held one of the 'Views from the Site' meetings in Birmingham on Thursday, 5 March 1981.

5 ITA meetings, Nice, May 1981

The Seventh Annual Meeting of the ITA will be held in Nice from 10-14 May 1981 in association with the AFTES Conference 'Cost cutting in Tunnelling'. Registration forms are available from the Secretariat.

6 Eighth ITA meeting, Brighton 1982

The Eighth Annual Meeting of the ITA will be hosted by the BTS and will be held in association with 'Tunnelling '82' at the Brighton Metropole from 7-11 June 1981.

7 The Engineering Group of the Geological Society

The Society has agreed to hold a joint meeting at the invitation of the Engineering Group of the Geological Society at Burlington House, Piccadilly, London. The probable title of the meeting is 'Predictions and reality in tunnel site investigations'.

8 Sewerage '81

Members may be interested in the Conference 'Sewerage '81' which is being organised by the Institution of Civil Engineers at the Kensington Exhibition Centre, London from 22-24 June 1981. Further information is available from the The Conference Office, Institution of Civil Engineers, 1-7 Great George Street, London SW1P 3AA (01-222 7722 ext. 261).

MINE DEVELOPMENT

Track Drifting With the Hagglund System

G. O'GORMAN, Underground Superintendent,
Corbet Mine, Falconbridge Copper Corporation,
Noranda, Quebec

Presented at the CIM Underground Operators Conference,
Sudbury, Ontario, February 1981

This paper discusses the experiences of Corporation Falconbridge Copper, Lake Dufault Division, Corbet Mine, with the Haggloader and Hagglund Car System for driving track drifts. A performance appraisal of the system in comparison to standard equipment will be made. Limitations will also be discussed.

Glenn O'Gorman was born in Matachewan, and raised in Malartic, Quebec. He attended Queen's University in Kingston, Ontario, and graduated in 1974 with a Bachelor of Science (mining). He worked for Heath Steele Mines Ltd. for five years in various positions and joined Falconbridge Copper in 1979 as production engineer. He is currently underground superintendent at the Corbet Mine.

Introduction

The Corbet orebody is located some 5 miles north of the city of Rouyn-Noranda, in the Abitibi greenstone belt, and is the third of three orebodies to be found by Lake Dufault (Fig. 1).

Corporation Falconbridge Copper began shaft sinking in the Lake Dufault area in 1962, and opened the Norbec Mine in 1964. Shortly thereafter, in 1966, the Millenbach deposit was found and brought into production in 1971. The Corbert orebody, found in 1974, came on-stream January 1, 1980. All three deposits are of the classic volcanogenic massive sulphide type, and are overlain by 1,100 to 2,200 feet of volcanic rock. The orebodies found to date have yielded approximately 8 to 10 years each of mill feed. Their small size, roughly 3.5 million tons each, as well as their depth, have required extensive underground, as well as surface, exploration programs. Our present underground program calls for in excess of 10,000 linear feet of underground exploration drifting per year, with drifts in excess of 1 mile long. Ninety per cent of these drifts are driven with track.

Background

Hagglund cars were first employed at the Millenbach Mine in 1974 with good results, hence it was decided to continue their use at the Corbet Mine for exploration headings.

The Hagglund system was developed in Sweden in 1954, and has since been used throughout the world. It is generally used for driving long headings, but it can be used to advantage in medium-range applications as well.

The "Hagg" System vs the "Standard" System

The traditional method of mucking track headings employs a rocker dump shovel and a train of cars. One car is loaded and then trammed to some sort of switching mechanism, where it is switched, and the train is returned to the face to load the next car. This process is repeated until each car in the train is loaded, and then the train is trammed to the dump location and dumped. Normally, the train capacity is less than the muckpile, so that two or more trips must be made between the face and the dump.

The Hagglund system employs a gathering-arm type of loader that feeds the muck onto a flight conveyor which dumps into 22-ton Hagglund cars. Each car is designed with a flight conveyor built into the floor plan of the car so that when the first is full, the muck can be transferred into a second car without having to switch cars. Depending on the circumstances, a train of these cars can be coupled together with a capacity equal to the actual muckpile. In essence, the train, comprised of loader and cars, leaves the dumping point, travels to the face, mucks out the round and returns to the dump point, complete with the loader and a complete round of muck.

Equipment — Hagglund System

Haggloader 8HR-EL

This loader operates by pushing muck into a flight conveyor by means of gathering arms. The flight conveyor then moves the muck back, and drops it into a car underneath. The loader which we are using is electrically powered by a 600-V system, and requires 50 hp to operate. The pneumatic model requires 1000 to 1200 cfm to operate, but otherwise is identical to the electrical model. The loader can operate in headings down to 7 ft wide by 7.5 ft high. Our drifts are driven 11 ft high by 9 ft wide to accommodate 30-in. vent tubing, and to provide sufficient clearance for safe operation. The loader can negotiate a 40-ft-radius curve, but requires a minimum of 100-ft-radius curve for mucking. Height in the transport position is 6 ft, 8 in.; however, the rear end of the conveyor is adjustable for loading into cars. The loading end of the conveyor is also adjustable, and can be lowered to 10 inches below the track. The unit can muck up to 10 ft ahead of the track, which eliminates the need to move the slide rails frequently during the mucking cycle; once per cycle is sufficient, and not always necessary.

The loading method of the 8HR eliminates the jerky motions common to overhead loaders, and improves safety, as there is no danger of muck falling off the conveyor and striking the operator, as with the overshot mucking machines. Operator acceptance was immediate, and very little training was required.

Hagglund Cars

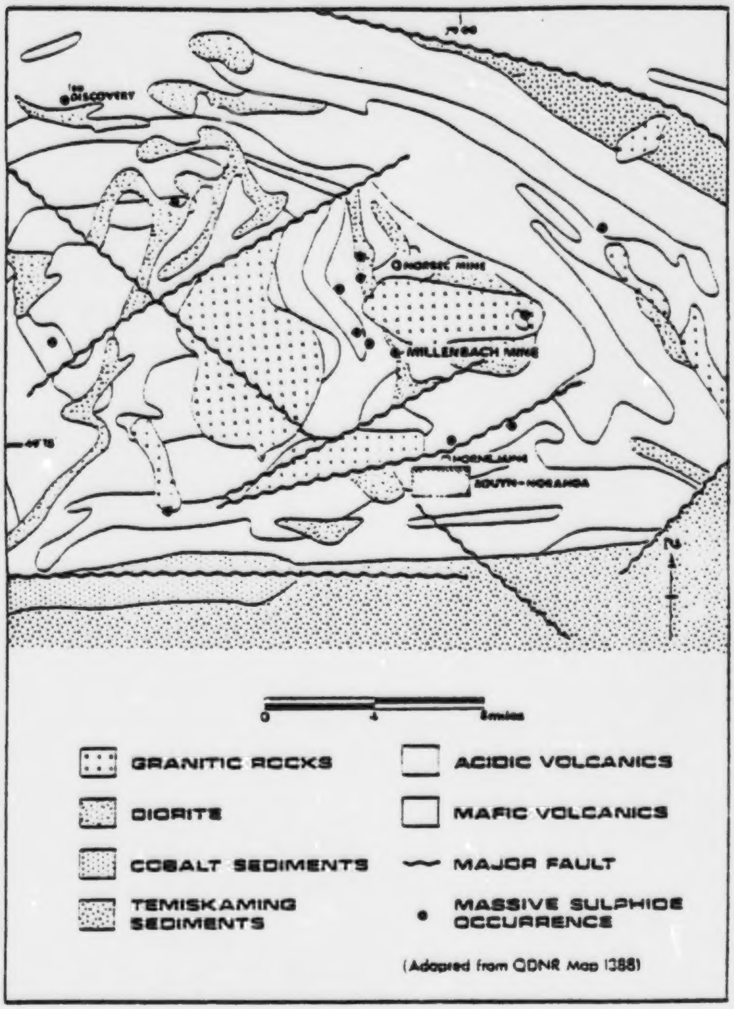
We are using 22-ton, model 115-B, 400-cu.-ft "Hagg" cars. These cars are designed with flight conveyors along the bottom of the car; the bottom of the car is sloped from front to back so that muck can be transferred from one car to the other.

Experience and Performance

Most people agree that an advance cycle in a development heading is basically composed of three time components, which are roughly equal in duration, namely: a) mucking, b) drilling, and c) auxiliary operations such as loading the round, laying track, installing pipe and blowing ditch. When any one of the above three operations begins to require more than one-third of the available productive time, any crew will begin to miss rounds. Generally speaking, only the mucking cycle is dependent on the distance between the face and the dump point.

For purposes of this paper, elements b) and c) are considered to be constant, although we did try to improve the drilling aspect with the use of a two-boom jumbo. It was found, however, that the size of the jumbo itself was too unwieldly in a moderate-sized single heading.

In order to keep the mucking cycle time to a minimum, we purchased a Haggloader 8HR-EL for use with Hagglund cars in our third-level exploration drift. The primary aim of purchasing the Haggloader was to cut down on the actual mucking time at the face.



(Adapted from QDNR Map 1388)

FIGURE 1. Location map and general geology, Noranda area.

Our "standard" equipment for mucking has been the Atlas-Copco LM-56 overhead loader. The predicted mucking capacity of the 8HR was 71 cu.ft per minute, as compared to 35 cu.ft per minute for an LM-56. Given a drift round 11 ft high by 9 ft wide, with an anticipated advance of 8.5 ft per. round, and assuming 22% overbreak and a swell factor of 50%, the volume of muck to be loaded per round is 1,540 cu.ft. Thus, the theoretical mucking time for the 8HR is 21.7 minutes per round, as compared to 44 minutes' mucking time with the LM-56. This constitutes a saving of 22.3 minutes on the mucking cycle at the outset. When one considers that in an eight-hour shift, 30 minutes are required for lunch, then there remain only 2.5 hours for each of the three elements of the drifting cycle. Thus, the 22.3 minutes' saving in mucking time represents a substantial 14.8% of the available time for the mucking cycle.

Regardless of the type of equipment used to muck in a track drift, certain elements of the mucking cycle are fixed, regardless of the distance. These elements are: 1) collect locomotive, cars and loader (3 min); 2) connect to airline and daily checks (5 min); 3) flyrock cleanup (4 min); 4) dump cars (12 min) — this is an actual dumping time for either the Hagglund cars or Granby-type cars for one round—; 5) disconnect train and loader, and park (5 min). The total time required for the fixed elements is 29 minutes, which leaves only 2 hours and 1 minute for actual mucking and tramming to take place. Thus, the aforementioned 22.3 minutes' saving in loading the cars now represents a time saving of 18.4%. To put it another way, mucking with the Haggloader requires only 21.7 minutes of the available 121 minutes, whereas mucking with the LM-56 requires 44 minutes, leaving 99.3 minutes and 77 minutes, respectively, for tramming, travelling to the face at the start of the shift and switching cars.

Obviously, any system which can reduce the number of trips between the face and the dump point, or the total time required to switch cars, will be advantageous.

At the Corbet Mine, we have employed combinations of the Haggloader and Hagglund cars, LM-56 muck machines with three Granby cars and a side switch at 500-foot intervals, two locos, each with two Granby cars, and side switches at 500-ft intervals, and an LM-56 with one Hagglund car. In all cases, the drifts are the same dimensions, and the drilling equipment consists of a rail-bound drill platform with a three-man crew.

At the time of writing, we were using the Haggloader 8HR-EL and two Hagglund 115-B cars, each with a capacity of 400 cu.ft. The distance from the face to the dump point is 3,000 ft. The volume of muck is slightly more than our previously calculated 1,540 cu.ft, so that it requires 4.25 Hagglund cars per round.

Recalling that there are 121 minutes allowed for variable elements of the mucking cycle, we find that the round has been mucked out with 50 minutes to spare. Including all fixed and variable times, the total mucking cycle has required only 1 hour, 30 minutes and 9 seconds at the 3,000-ft distance.

Our performance in this heading reflects the extra available time in that we have obtained an advance of 3.22 ft per manshift over the past month at approximately 2,700 to 3,000 ft from the dump point with a three-man crew.

In another case, we found it very difficult to take one round per shift at a distance of 1,470 ft from the dump point using three 90-cu.-ft Granby cars, and a side switch at 500-ft intervals, so we removed the Granby cars and replaced them with one 400-cu.-ft Hagglund car. The performance per manshift went from 2.32 ft to 2.93 ft (average) over the following six weeks. Although we were mucking with an LM-56 as opposed to the Haggloader, the extra capacity of the Hagg car, along with the elimination of the requirement to switch cars, saved a considerable amount of time. The three 90-cu.-ft Granby cars required more than the previously mentioned 121 minutes of available variable time to muck out the round, and the crew started to miss rounds.

A third case is our 15th level, where we are using two small locos, each with two 90-cu.-ft Granby cars, at 1,300 ft from the dump point. The performance has been fairly good up to now, because the tramming time was greatly offset by having the two locos, because when one loco was gone to the dump, the other was filling its two cars. This system, however, has essentially reached its limit and the crew has begun to miss rounds.

Cost Analysis

One Hagg car costs $108,000, whereas a 90-cu.-ft Granby type can be obtained for $6,000. However, it requires 5 Granbys per train for a cost of $30,000. Locos are roughly the same size, due to payload. If we base the cost on over-all cost to drive a 5,000-ft drift, we must also include the cost of cutting switches to transfer cars. We have found that it costs approximately $5,000 to cut a siding to switch cars. Normally, we cut sidings at 500-ft intervals, so the over-all cost for 5,000 ft of drift is $45,000.

The ability of the crew to take a round per shift at long distances from the dump point is the real cost advantage. Consider that the crew, performing at 2.32 ft per manshift, is able to advance at the rate of 2.93 ft per manshift simply by switching from three 90-cu.-ft Granby cars to one Hagg car, while their bonus remains relatively the same, and the cost advantage translates into a savings of $13.50/ft, although a 1,500-ft drift can be driven more cheaply by means of Granby cars. At an average cost savings of $13.50/ft for 5,000 ft, this amounts to $67,500.

The extra advance per year that can be achieved with the Hagg System is difficult to evaluate monetarily, but in some circumstances speed may be a governing criterion. If we use the extreme case from the above examples, the difference between 3.22 and 2.32 ft per manshift is 0.9 ft per manshift. At 220 working days per year, this translates into 1,188 ft per year at two shifts per day. With conventional equipment, it is not likely that much more advance could be made by going to a third shift because of reduced efficiency. Again, depending on circumstances, the cost of a third shift where normal operations are carried out on a two-shift basis could be more than the cost of the Hagg equipment over a period of less than two years.

Although the cost savings using a Haggloader are not expected to significantly exceed its $146,000 price tag, it has considerable advantages over the overshot loader with regard to operator safety and fatigue.

Advantages & Limitations

As previously mentioned, the Haggloader has a number of safety features which make it desirable. The machine does not employ the overshot technique, with the danger of the operator being injured by a piece of muck falling out of the bucket, nor does it have the jerky motions of an overhead loader. It operates smoothly and continuously. As an added feature, water sprays are installed on the digging arms so that the muckpile is being continuously washed while mucking, and dust exposure is minimal.

The Hagg system is especially good for extremely long, straight tunnels. At medium range— distances of 2,000 to 4,500 ft — only two Hagglund cars are really necessary, as the time during which the Hagglund cars are travelling to and from the dump point at this distance can efficiently be made use of to scale the back, hang pipe and vent tube, blow ditch and prepare to lay track.

There are, however, limitations with the system. One of those is the price tag. They are very expensive pieces of equipment. Maintenance on the Hagglund cars has not caused any problems, and in fact has been excellent, with fewer breakdowns than with conventional equipment. The electric trailing cable on the Haggloader has been the cause of most of our delays. Great care must be taken not to snag or twist the cable or damage it in any way. We have overcome the majority of the problems with the cable by a) reducing the length of the trailing cable to less the 300 ft, b) hanging the cable on pins along the wall to keep it out of the water in the ditch (a major cause of shorting), and c) keeping a second duplicate cable so that if there is a problem with one cable, the crew unplugs the problem cable and replaces it with one which has previously been repaired.

The Haggloader can only load in curves of over 100-ft radius, and transferring between cars is limited to curves where the radius exceeds 265 ft for B-type cars. Thus, any proposed drift must be as straight as possible, unless C-type cars are used.

Conclusions & Recommendations

The Hagglund cars have certainly proven their worth at our operation, and are to be seriously considered whenever one has to drive a drift in excess of 2,000 ft. The Haggloader 8HR-EL has proven to be satisfactory. However, at any operation which has sufficient compressor capacity, the air-driven model would be preferable.

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ASEA

Pamphlet AS 20-114 E Edition 1

Computer-controlled ore transportation at the Kiirunavaara iron ore mine in Kiruna, Sweden

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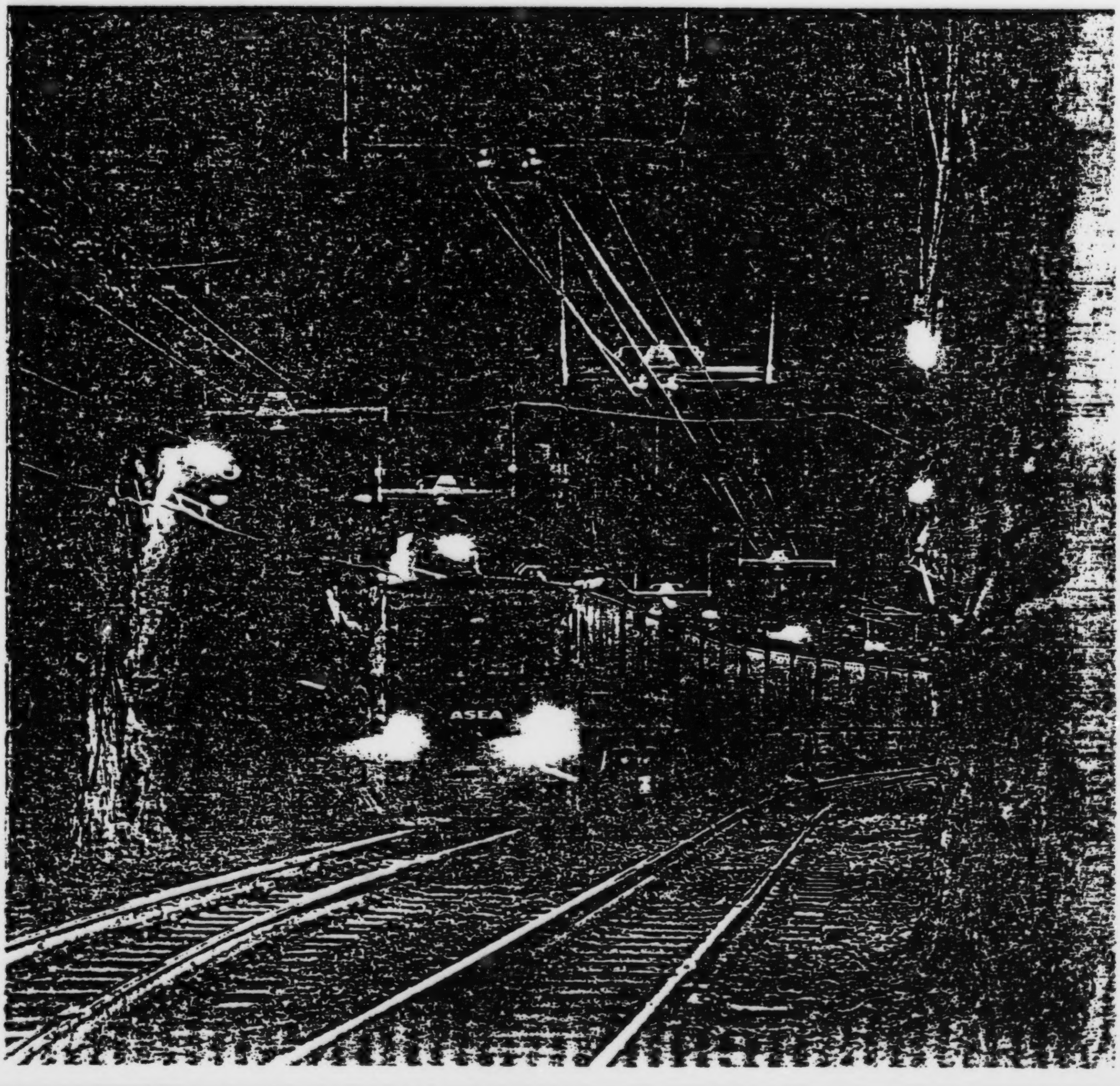

ASEA

Computer-controlled ore transportation at the Kiirunavaara iron ore mine in Kiruna, Sweden

Bertil Oberg, Process Industries Division

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ASEA Reg. 573, 6131, 6152, 752

Reprint from ASEA Journal 1980:3, pp. 35–40

Description of a highly automated transportation system supplied by ASEA to the Kiirunavaara Iron Ore Mine for a new main haulage level at 775 metres, with special emphasis on the train haulage, locomotives, crushing plant, main functions of the control system, train control, crusher control, computer system and control modes. The experience gained during the commissioning of the transportation system has confirmed the advantages of the decentralised ATC system and the hierarchical structure of the control system.

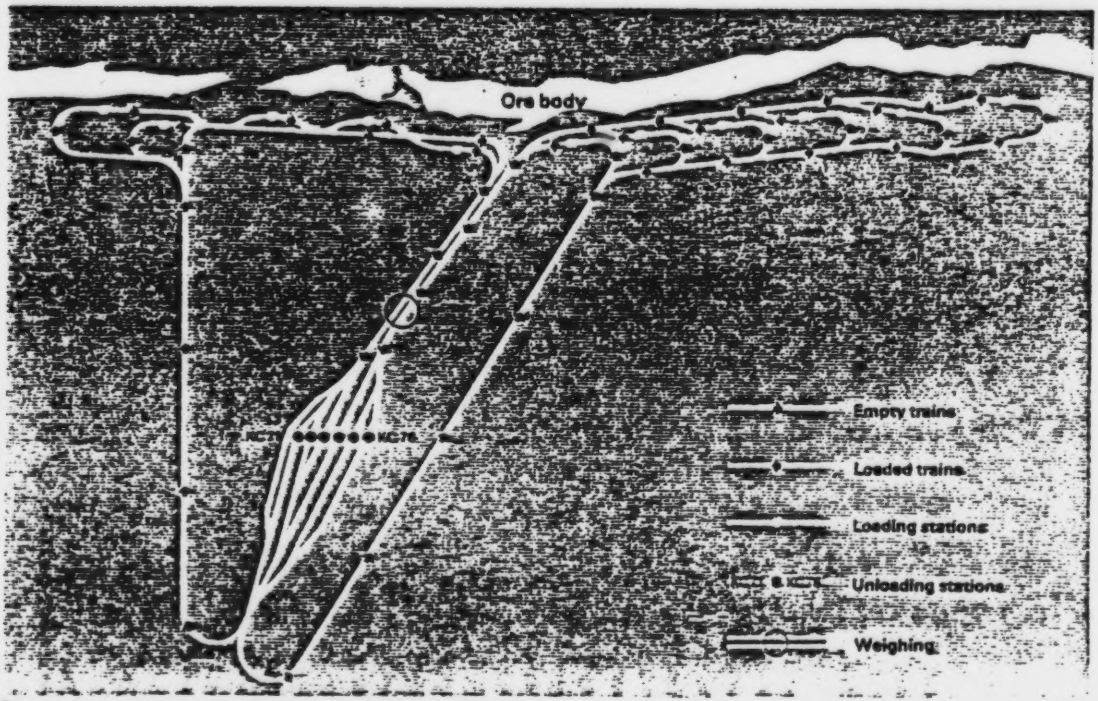
The Kiirunavaara Iron Ore Mine, located at Kiruna north of the Arctic Circle, in Swedish Lapland, is one of the largest underground mines in the world. It belongs to the State-owned company Luossavaara-Kiirunavaara AB (LKAB). The mine is one of the most advanced as regards mining and transportation technology.

Last year LKAB started up a highly automated transportation system for a new main haulage level at 775 m. This project, known as KUJ 75, has a design capacity of 30 million tonnes per year. An overall computer system supplied by ASEA controls the flow of the ore from the chutes via a train haulage system and a crushing plant to the measuring pockets of the hoisting plant.

The ore body of the Kiirunavaara Mine is tabular and has an inclination of 40 to 75°. It has a length of about 4 km and an average thickness of 90 m. The ore body consists of interbedded low-phosphorus and high-phosphorus ores.

Ore is transported on the working levels to a series of gravity shafts along

Fig. 1. Track layout on the 775 m level in the LKAB mine at Kiruna.



the ore body. When the KUJ 75 project has reached full production, there will be up to 51 gravity shafts with chutes on the haulage level, arranged in 13 loading galleries (see Fig. 1). From the gravity shafts the ore is transported on the 775 m level by trains to the central crushing plant. Six unloading stations and crushers will be installed for the dumping of the train loads.

Control of the transportation on the 775 m level is part of a master system controlling all production flow, from the ore body to the processing plant. One of the major tasks of the control system is to separate the flows of ore with different phosphorus contents and to control the desired blending of the grades.

Train haulage

Each production train is made up of one ASEA 65-tonne thyristor chopper-controlled locomotive, type LGTB 65, and 18 bottom-dumping cars (see Fig. 2). The average train load is 460 tonnes. The control system is designed for simultaneous operation of 25 unmanned production trains and five service trains.

Loading of the trains from the chutes is remote-operated from the central control room by means of closed-circuit television. During loading a sample of the ore is sent via a pneumatic tube system to a laboratory for phosphorus analysis.

The train load is automatically dumped while the train is passing one of the unloading stations.

Locomotives

The ASEA locomotives are of 4-axle bogie type and their maximum speed is 25 km/h. Each of the four traction motors has a continuous rating of 93 kW. The trolley line voltage is 600 V, d.c.

The four parallel-connected, separately excited traction motors are normally supplied by a thyristor chopper either from the trolley or, when the locomotive is in the loading and unloading areas, from a traction battery. In the event of a failure in the control circuit this traction battery can be used to feed the traction motors directly. A fully loaded train can then be removed to a location where it does not interfere with other traffic.

The locomotive uses rheostatic braking with full braking effort down to a low speed, when pneumatic disc brakes automatically take over. The control system includes target braking equipment, which gives a constant braking distance, irrespective of the initial speed and the train load.

As mentioned earlier, the locomotive is designed for unmanned operation, but it also has a driver's cab to permit convenient manual operation.

Crushing plant

Six crushers are being installed in the underground crushing plant underneath the haulage level. The crushed ore is transported through a system of chutes and conveyors to ten bins and, further, to the measuring pockets serving the ten hoists. This conveyor system is highly flexible, allowing a large number of alternative transport routes to be selected.

Control system functions

The control system is built up in a hierarchical way from the following subsystems (see Fig. 5):

- Dual computer system
- Data transmission system
- Automatic train control system (ATC)
- Safety signalling system
- Remote control of loading
- Logic controller for the crushing plant

The control system includes the following main functions:

- Train control
- Remote train loading
- Crushing plant control
- Supervision of trolley system

The system functions are designed to make control possible with different degrees of sophistication.

Train control

Safety signalling system

The train control is based on a safety signalling system using track circuits for the train detection (see Fig. 3). This system has been supplied by L.M. Ericsson acting as subcontractor to ASEA.

The rails are electrically insulated in sections or blocks. A current source and a relay are connected between the two rails of each block. The relay is energised as long as no train is in the block. When a train enters the block, the relay is short-circuited and de-energised by the train wheel sets.

The safety signalling system interlocks the operation of the switch points and operates the signal lights. Information about occupancy of the track circuits, signal state and position of switch points is also transmitted to a mimic diagram in the control centre and to the computer system. The safety signalling system also interlocks directly speed commands in the ATC system.

The lowest degree of sophistication of the train control would be to use the safety signalling system for Centralised Traffic Control (CTC), with manned trains.

Data transmission

L.M. Ericsson has also supplied a data transmission system, type JZA 700, which is integrated with the safety signalling system. The transmission system is used for communication between the control centre and the track, i.e., track circuits, signals and switch points. Commands can be given from both the computer system and separate keyboards.

The data transmission system is also used to control and monitor the trolley system and for communication with weighing and sampling equipment.

To ensure a high availability, the data transmission system has been divided into three separate subsystems, one for each of the two main track loops and one for communication with the equipment in the control centre.

ATC system

The ATC system has been supplied by SAAB-SCANIA acting as subcontractor to ASEA. The system is decentralised and directly interlocked with the safety signalling system.

An inductive transmission system, type SAAB MDT, is used to transmit commands to the locomotives (see Fig. 6). Transmission antenna loops are mounted along the track above the trolley wire or between the rails. The antenna, consisting of a single-core cable, is fed through isolating transformers to protect the transmission equipment from overvoltages in the event of the antenna coming into contact with the trolley wire.

The transmission system conveys commands to the locomotives as continuously repeated 16-bit messages transmitted serially by means of frequency-shift keying at a centre frequency of 30 kHz.

Each locomotive is provided with two pick-up antennas for antenna loops overhead and on the track bed, respectively. The command telegram received is carefully checked and decoded and the commands are forwarded to the locomotive control equipment. If the telegram is not continuously received or if it is not accepted during the checking.

ASEA

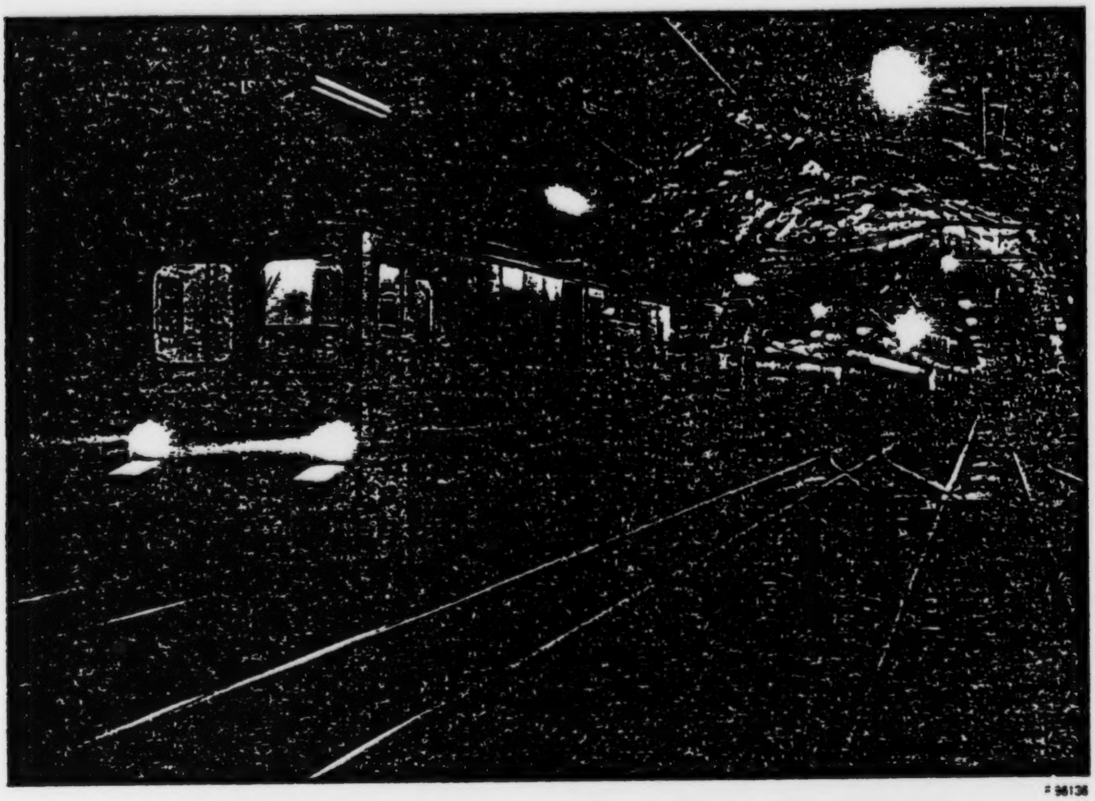
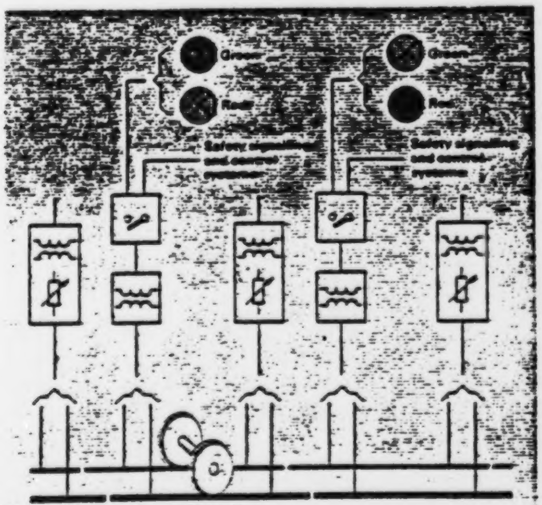


Fig. 2. An ASEA 65-tonne thyristor chopper-controlled locomotive type LGTB 65.

Fig. 3. Track circuit system.

Fig. 4. Control panel for underground crushing plant.


Green
Red
Safety signalling and control systems
Green
Red
Safety signalling and control systems

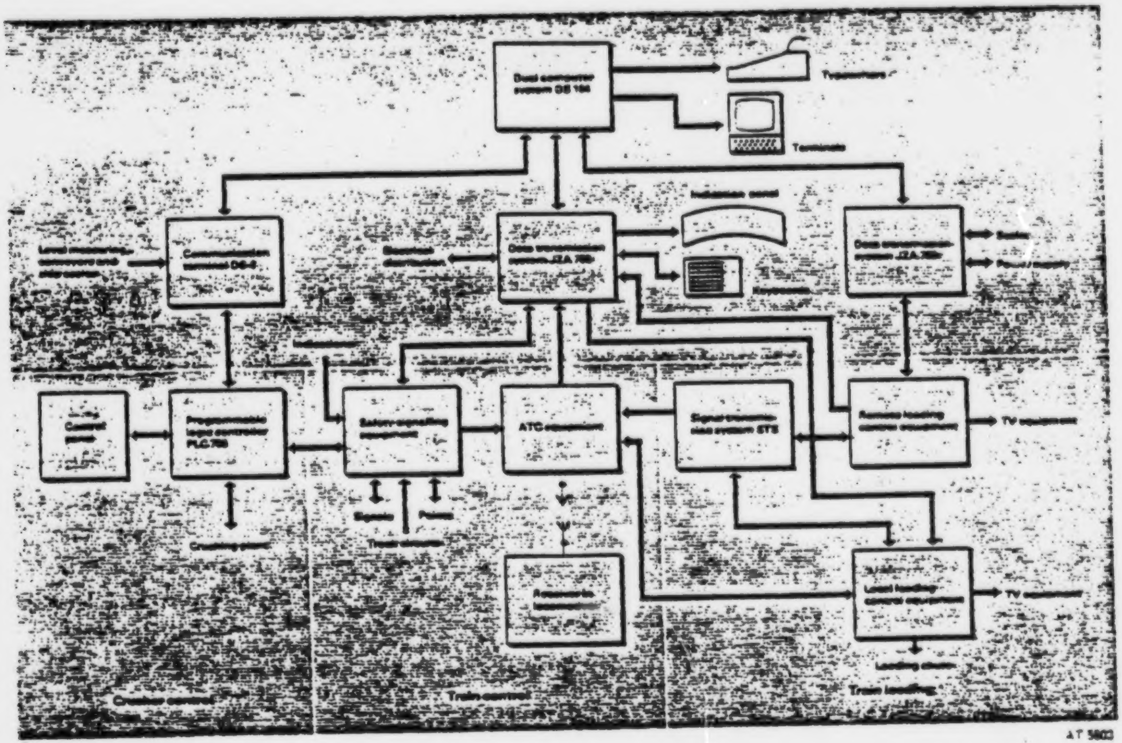

Typewriters
Terminals
Indication panel
Programmable logic controller PLC 700
Control panel
Safety-signalling equipment
ATC equipment
Signal transmission system STS
Remote loading control equipment
TV equipment
Local loading control equipment
Loading chute
Signals
Points
Track circuits
Crushing plant
Crusher control
Train control
Train loading
AT 5603

Fig. 5. Computerised control of train haulage and crushing plant in the LKAB mine at Kiruna.

emergency braking of the locomotive is ordered.

The transmission antenna is divided into loops corresponding to the track circuit blocks of the safety signalling system. An antenna loop is connected to the transmitter only when the safety signalling system indicates that a train is located in the corresponding block. This assures that all train movements are supervised.

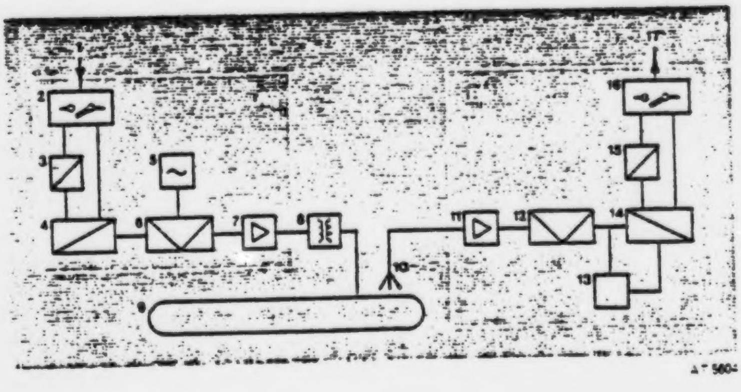
The train speed level ordered from the ATC equipment depends on the number of free blocks ahead of the train. If there are two free blocks, the maximum speed programmed for the track section in question is ordered. In front of the track signals there is a separate antenna loop, the length of which is adapted to the braking distance of the train. If the signal is red, a command for zero speed is transmitted. This command applies target braking of the train. Two different creep speeds, forward and reverse, can be ordered for trains in the loading areas.

A route for a desired train destination is set up by means of the safety signalling system. The information from the track circuits to the ATC system initiates the transmission of speed commands to the locomotive. The direct connection between the safety signalling system and the ATC system facilitates the com-

Fig. 6 Inductive transmission system type SAAB MDT for train control.

A Stationary transmitter
B Receiver in locomotive

1 ATC commands
2 Input relays
3 Binary/decimal encoder
4 Parallel/series convertor
5 Oscillator
6 Frequency-shift keying modulator
7 Amplifier
8 Line transformer
9 Loop antenna
10 Receiver antenna
11 Amplifier
12 Frequency-shift keying detector
13 Receiver check logic
14 Series/parallel convertor
15 Binary/decimal decoder
16 Output relays
17 To locomotive control equipment


1
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AT 5604

missioning and makes the automatic train control very reliable.

Crusher control

The crushing plant is also provided with a decentralised control system, using six ASEA PLC 700 programmable logic controllers. An instruction for a certain route through the crushing plant can be ordered from the computer system or from a local control panel (see Fig. 4). ASEA's DS-8 system is used for communication between the local crusher control and the computer system.

Computer system

The computer system is built up from two DS 104 computers (see ASEA Journal 1978:3, pp. 78–79). Using a dual computer system gives the following advantages:

- High availability
- Programs can be developed and modified without the risk of disturbing the on-line functions
- System maintenance can be performed without taking the computer system out of operation

Typewriters and two types of visual display unit are available for the operator's communication (see Fig. 7). Alphanumeric display units and keyboards are used for commands and reports, while multicolour semigraphic display units are used to supervise the crushing plant and the computer system configuration. It is possible to choose between overview displays and detailed displays for different parts of the plant.

Control modes

Computer control of trains

In the normal control mode the computer system is used for the overall control of the ore flow. Certain priority plans for the loading chutes from the higher-level production control system are incorporated in the KUJ 75 control system.

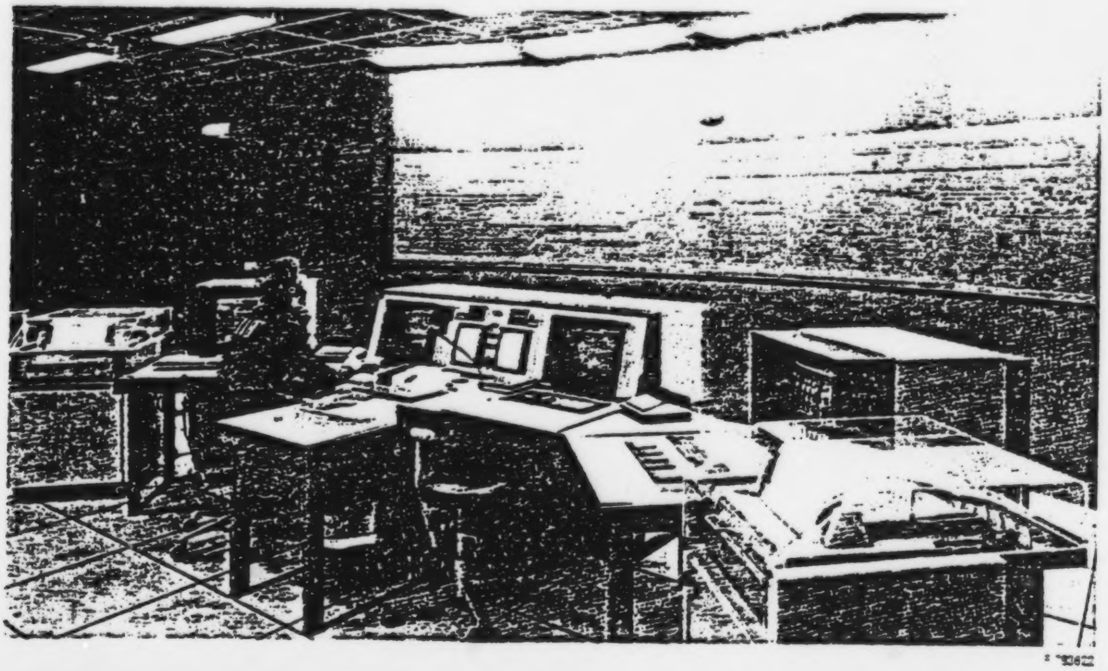
When an empty train leaves the unloading area, the system selects, according to the priority plan, a suitable station for the loading of the train as well as the best route to this destination. The system then gives successive orders to the safety signalling system for route reservation up to the loading station. When a reservation has been executed, a speed command is transmitted to the locomotive by the ATC equipment. The computer system continuously supervises the movement of the train to assure that it is following the selected route. If the selected loading station is blocked or if any disturbances occur along the selected route during the movement of the train, the system automatically changes the destination or the route of the train.

When a train has arrived at a loading station, this is announced on the seven operating desks for remote loading (see Fig. 8). One of the loading operators takes over the command of the train. The TV camera at the loading station is automatically connected to the monitor on the desk. The operator controls the chute mechanism and the movement of the train, observing the loading with the aid of the TV display. When the train is loaded, he gives a signal to the computer system for taking over the control.

The computer system then has to direct the train to one of the six unloading stations. The selection of the unloading station depends on the grade of the loaded ore.

A preliminary destination of the train is chosen with regard to the previous phosphorus analysis of the ore from the loading station in question, a route reservation is made and the train starts. The ore sample that is automatically taken and sent to the laboratory is manually analysed. The result is entered into the computer system on a local computer terminal. The preliminary destination is checked and changed if necessary.

Fig. 7. Main control desk with typewriters and VDU terminals.



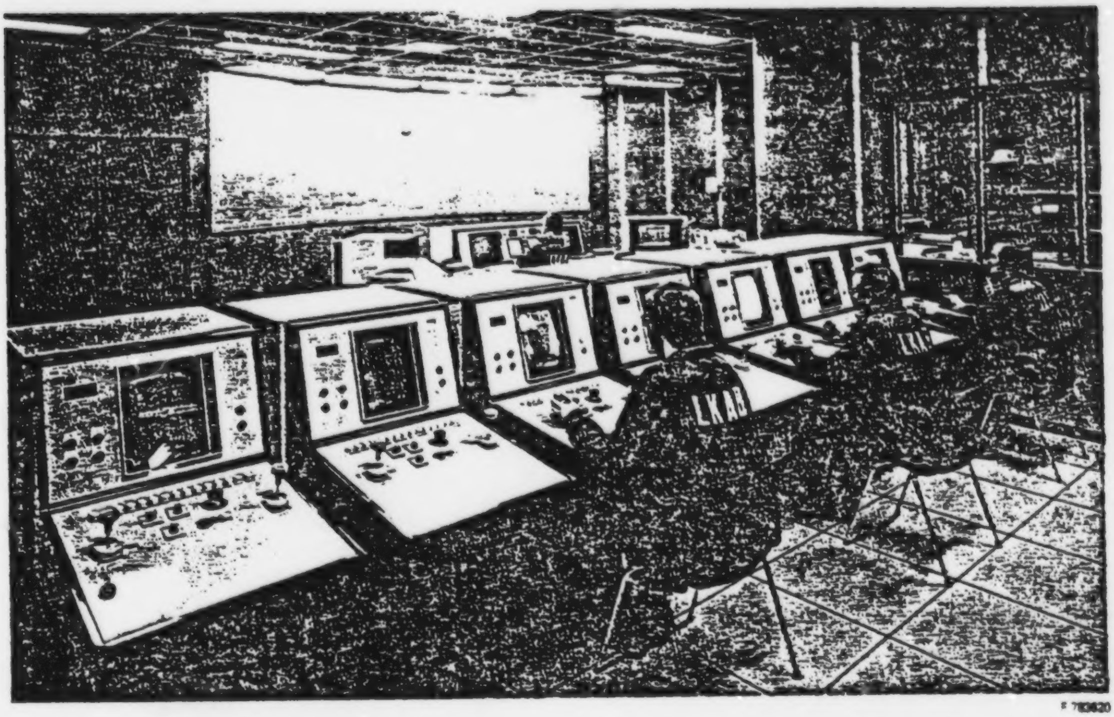


Fig. 8. Remote control of train loading.

Before reaching the unloading station, the train passes automatic weighing equipment, which feeds the weight of the load into the computer system. After passing the automatic unloading station for the bottom-dumping cars, the train is routed once more to a loading station.

The flow of ore through the crushing plant is directed with regard to the ore grade. The computer system selects the best route, also taking into account the amount of ore in the crusher bins, possible disturbances in the conveyor system, etc. The comand for a certain route is sent to the crusher control equipment for the execution of commands for the separate components in the conveyor system.

The state of the crushing plant is displayed on the local control panel as well as on the semigraphic display unit of the main control desk.

In addition, the computer system checks the state of the trolley system with rectifier substations.

Manual train dispatching mode

The unmanned trains can also be operated without the computer system. In such a case the train route reservation is ordered by means of a keyboard connected to the JZA 700 data transmission system.

Remote loading of the train is performed in the same manner as in the computer control mode. The routes through the crushing plant are selected on the local operating panel.

Conclusions

LKAB has long experience of train traffic control. A computerised control system for manned train traffic was put into operation at the 420 m level in 1965 and the 540 m level is operated with computer-controlled unmanned trains. The experience gained by LKAB has greatly contributed to the system design.

Experience from the commissioning and start-up of the KUJ 75 system has proved the advantages of the decentralised ATC system and hierarchical layout of the control system. The transport system will be successively expanded to its full capacity and full production over a period of about three years. The control of new track areas, loading stations and crushers can be easily implemented as required in the system already in operation.

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Low profile trucks keep underground costs down

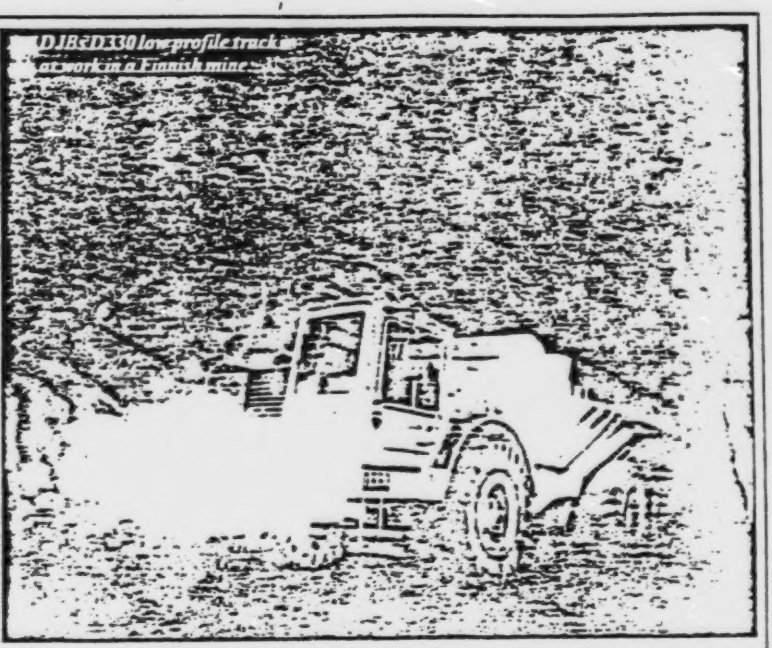
David Martin, Editor

A visit to the Vammala nickel mine, 30km south of Tampere in south west Finland, provided a opportunity to observe the effectiveness of a fleet of three British manufactured DJB D330 LP low profile articulated dump trucks in underground operation. The contractor, Lastause K'a'rj'A' & Pojat, is using the DJBs to transport rock and ore out of the mine, hauling the load a distance of approximately 2km, about half of the journey underground, up to a maximum drift grade of 1 in 8, reducing to 1 in 10 on the bends.

Shuttle service

The mine, owned by Finland's largest mining company, Outokumpu Oy, has an annual production of some 320 000 tonnes of ore, with about another 150 000 tonnes of waste rock to be removed as well. It was interesting to notice in passing that an old Tamrock L400 pneumatic 3-boom drilling jumbo was in use. The mine operates a 5 day week, 2 shifts/day system, using sublevel stoping with 25m between sublevel drifts as the mining method. Draw point loading is used to fill the trucks which run a shuttle service with normally three in use at a time, but sometimes one is withdrawn for maintenance. Trucks can pass inside the mine at suitably located special widened passing places. A CAT 980 is used to load the trucks, which then drive out climbing the ramp in 2nd gear when loaded (20km/h), returning empty and descending in 3rd gear (30km/h). The complete cycle time for one round trip is 35 minutes with the number of cycles per shift averaging 12. Up to 1000 tonnes can be moved in one shift when all three units are in use.

The low profile articulated DJBs have proved very effective at this site. The trucks are powerful enough to cope with the grades, and the articulation makes for better manoeuvrability and a sharper turning circle in the restricted space of the mine. The D330 LP has a payload of 30 tonnes making it possible to operate with the minimum number of units. It would obviously be a great disadvantage to have say twice as many vehicles carrying only half the payload each, and this proved to be one of the main attractions in choosing the high capacity DJBs.


DJB's D330 low profile truck at work in a Finnish mine

There are three different tunnel profiles inside the mine. The largest roadways are 4.8m wide at the base (Fig 1a) with the 1m diameter ventilation duct in the raised angle of the roof, the smallest (connecting) tunnels are 3.5m wide (Fig 1c) and there are also development tunnels which are 3.8m wide with the ventilation duct in the centre peak of the roof (Fig 1b). The DJBs can operate in all three without difficulty.

The three DJB330 LPs at this mine have the standard body with a low profile tractor. Two machines have been on site for just over 2 years and the third has been there for one and a half years. They each have a 30 tonnes payload with heavy duty body and high ducktail for superior load retention. Wide angles give easy load ejection. The LP versions have a low overall height, achieved by positioning the cab forward of the front axles. Dimensions are: height, only 2.4m; width 2.8m; and length, 9.128m.

The articulated hydraulic steering gives exceptional manoeuvrability and reversibility. There are independent torsion-free front and rear frames. The Caterpillar engine has four forward and three reverse speeds with full powershift transmission giving a maximum forward speed of 40km/h, 25km/h in reverse. The six wheel suspension system absorbs haul road and loading shocks, and the four wheel drive ensures maximum mobility and gradeability.

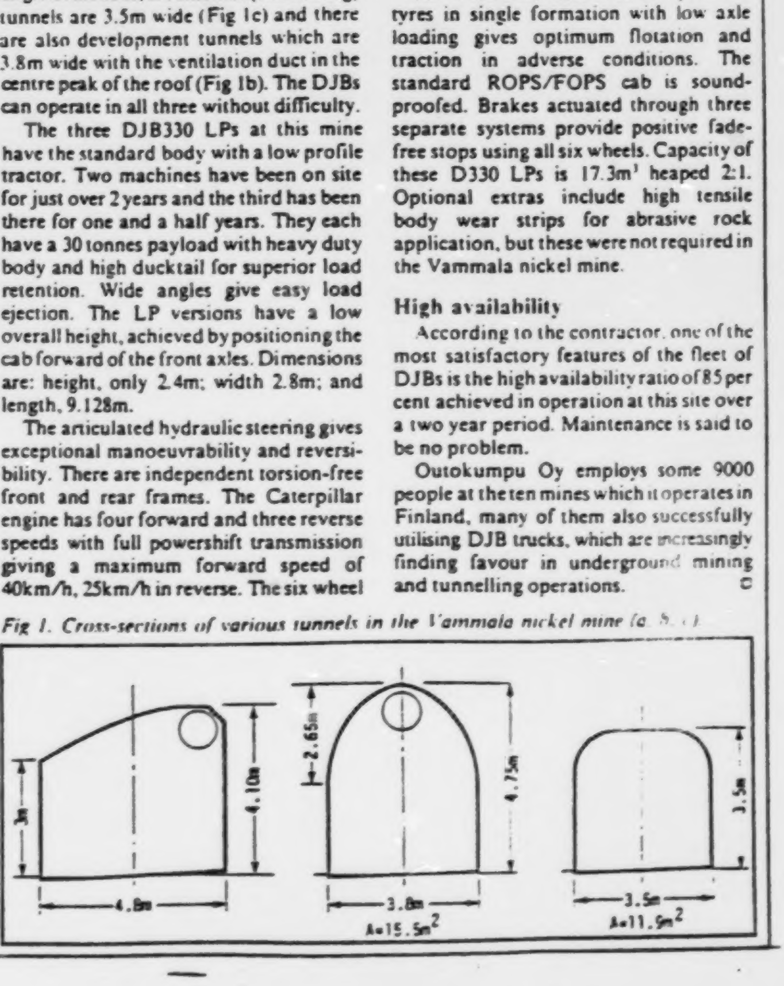
The wide base 23.5 x 25 low pressure tyres in single formation with low axle loading gives optimum flotation and traction in adverse conditions. The standard ROPS/FOPS cab is soundproofed. Brakes actuated through three separate systems provide positive fade-free stops using all six wheels. Capacity of these D330 LPs is 17.3m^3 heaped 2:1. Optional extras include high tensile body wear strips for abrasive rock application, but these were not required in the Vammala nickel mine.

High availability

According to the contractor, one of the most satisfactory features of the fleet of DJBs is the high availability ratio of 85 per cent achieved in operation at this site over a two year period. Maintenance is said to be no problem.

Outokumpu Oy employs some 9000 people at the ten mines which it operates in Finland, many of them also successfully utilising DJB trucks, which are increasingly finding favour in underground mining and tunnelling operations. □

Fig 1. Cross-sections of various tunnels in the Vammala nickel mine (a, b, c)



US BUREAU OF MINES DEVELOPS PORTABLE CRUSHER FOR UNDERGROUND HARDROCK MINES

Robert L. Schmidt, supervisory physical scientist, Twin Cities Research Center, US Bureau of Mines, Minneapolis, Minn.

A prototype, production-scale, mobile crusher for use underground has been developed under a US Bureau of Mines contract and successfully tested on the surface at a limestone quarry. Underground production testing is planned for 1982.

The need for the development of a mobile underground crusher arises from the fact that in underground hardrock mines, especially room-and-pillar mines, the highest-cost operation is usually the loading and hauling of run-of-mine ore. In medium- and large-scale room-and-pillar mines, according to one study, the materials-handling cost averages 32% of total operating costs.[1] The single most significant problem is the handling of oversize pieces, which can account for 15-25% of materials-handling manpower.[2]

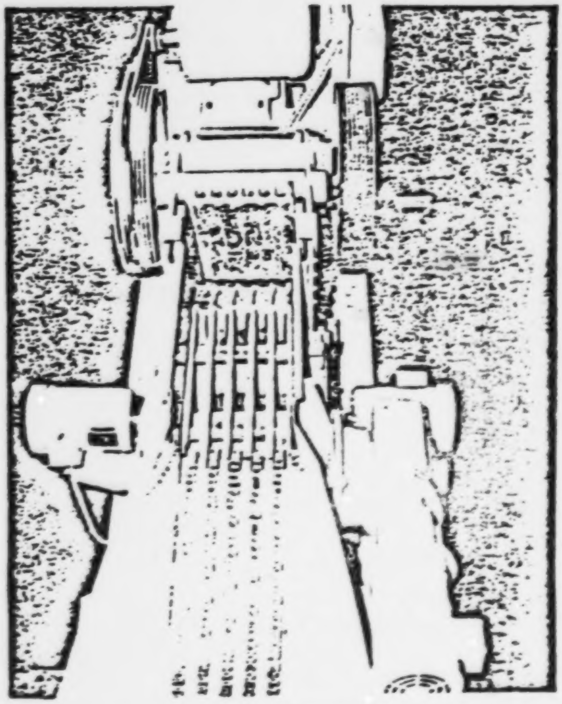
Conveyor belt haulage is generally accepted to be the most cost-effective system available for medium-distance horizontal transport. Conveyors are used effectively in coal and soft mineral mines, where the material size limitations imposed by such systems are relatively easy to overcome. If the extraction system does not adequately fragment the material, feeder-breakers can reduce the run-of-mine ore or rock to a size suitable for belt haulage. The feeder-breaker, being relatively portable, can be moved up as the mining face advances, so the belts can be extended.

Conveyor belts are also used in underground hardrock mines, but the nature of the material imposes restrictions. Run-of-mine ore is often blocky, with oversize pieces that make it unsuitable for feeding directly onto a belt. A common practice is to tram the run-of-mine ore a relatively long distance in a rubber-tired vehicle to a permanent underground crushing station. The crushed product is then transported by a belt. Because the ton-mile cost of conveyor systems can be as low as one-tenth the ton-mile cost of rubber-tired haulage[3], the ideal situation would be to have a portable hardrock crusher close to the face, feeding directly onto a conveyor belt.

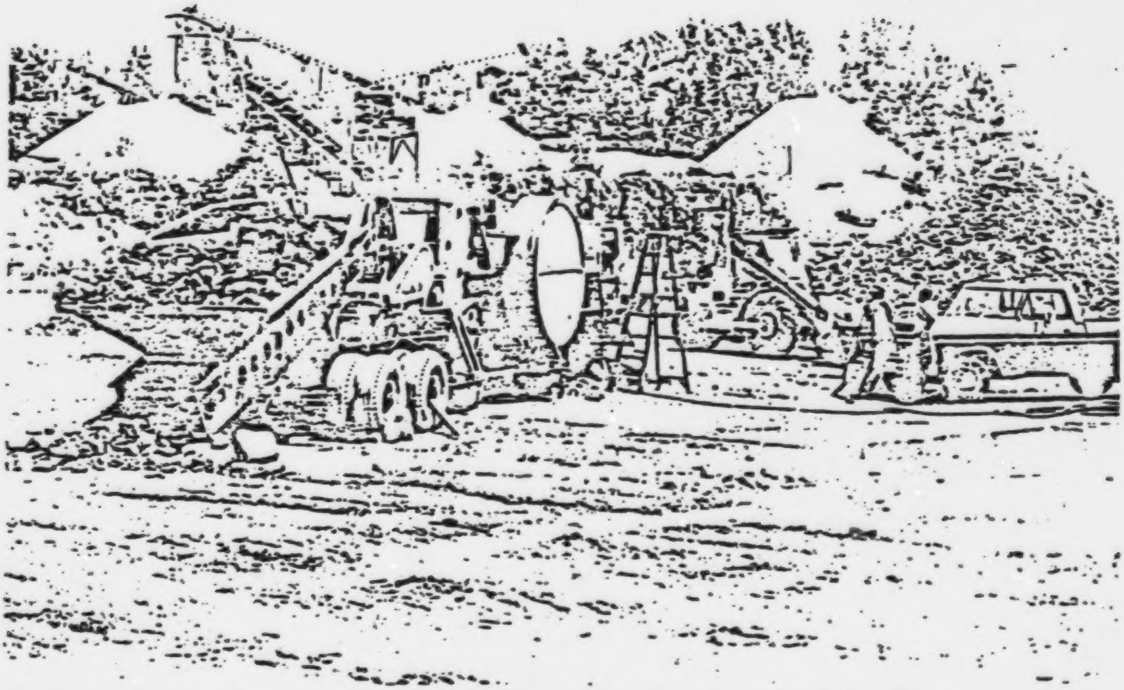
USBM DEVELOPMENT PROGRAM

In 1976, the Bureau of Mines initiated work to develop such a crusher. As with other equipment development efforts, the decision to proceed was made only after careful review. Bureau policy is to avoid competing with private industry, which would not be an appropriate expenditure of public funds. Rather, the Bureau sees its role as one of providing development funds in problem areas where private capital is not readily available and where a national need exists.

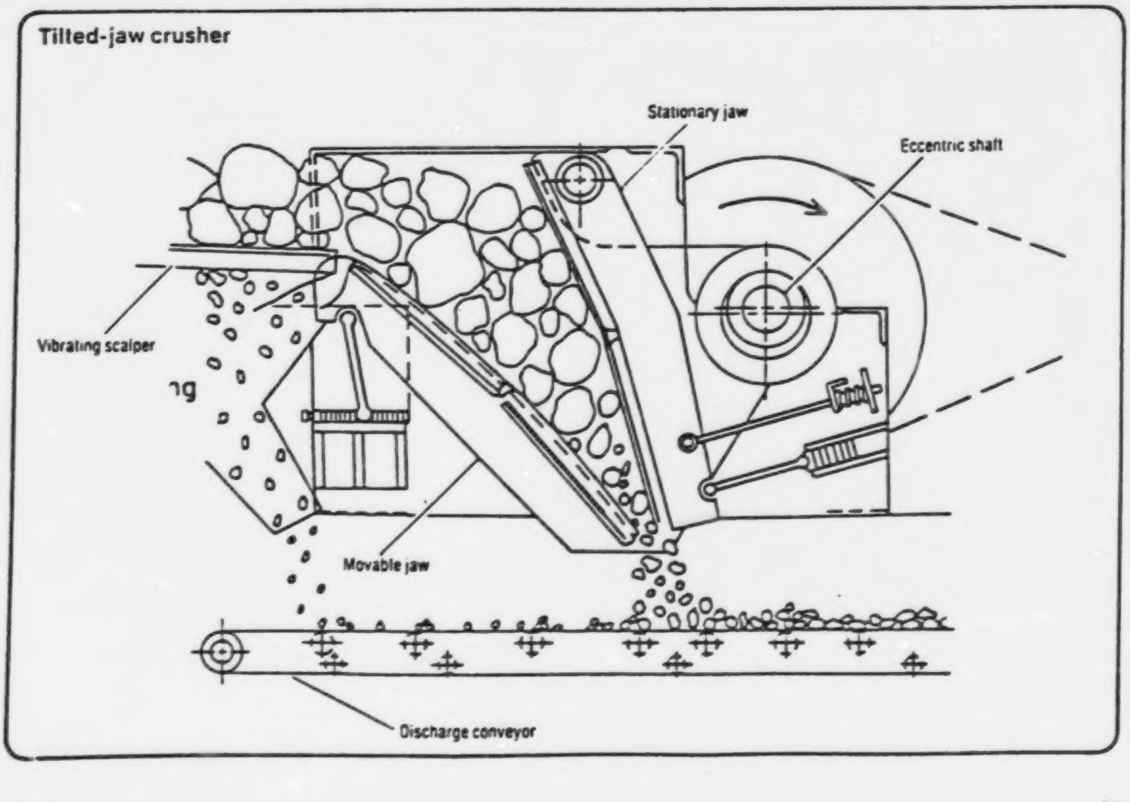
Development of equipment designed primarily for health and safety offers one example. Usually, the profit motive is lacking for such equipment, and the success of developments such as the low-dust continuous mining machine probably



The quarter-scale working model of the tilted-jaw crusher provided data for scale-up to a production-size unit.

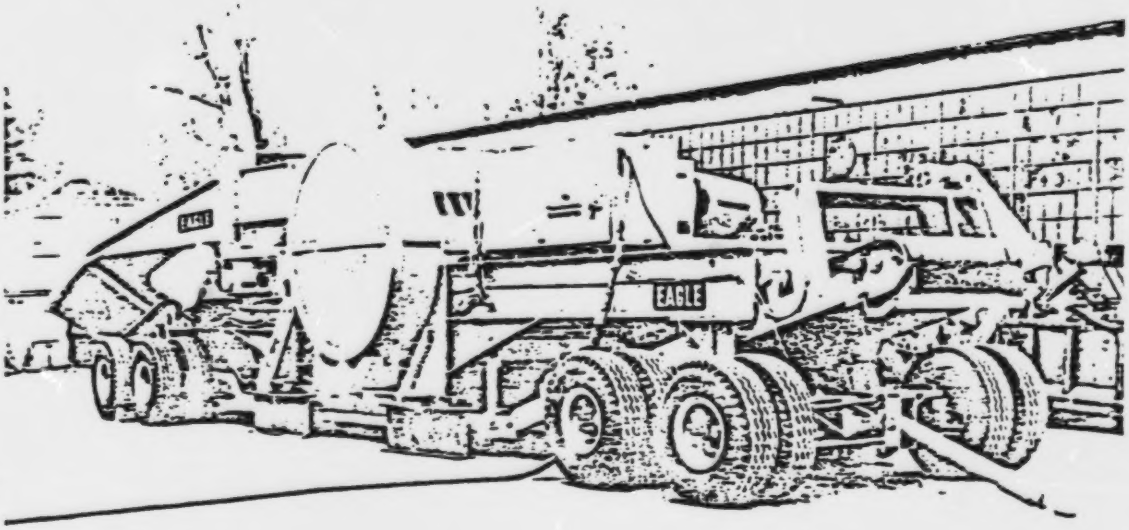


The prototype production-size tilted-jaw crusher underwent surface testing at the McCarthy limestone quarry in Upper Sandusky, Ohio.



Tilted-jaw crusher

Stationary jaw
Eccentric shaft
Vibrating scalper
Movable jaw
Discharge conveyor



At the Eagle Crusher Co. plant, the portable tilted-jaw crusher was incorporated into a system that includes an apron feeder, a vibrating grizzly, and a discharge conveyor, as well as the crusher, and the system was mounted on a 16-wheel trailer.

would not be achieved, were it not for government funding.

The use of government development funds is also indicated if the national need is apparent but the investment risks are unacceptable for the private sector. This situation occurs often in the mining machinery industry, where manufacturers tend to be small and have limited funds for research and development. This rationale has led to Bureau involvement in the large-diameter shaft drill, the retractable diamond core bit, and now, the portable underground crusher.

Having determined the research need, the Bureau next awarded an application study contract to Rapidex Inc.[3], which was subsequently acquired by Foster-Miller Inc. The purpose of the study was to define the potential market for a portable underground crusher and to determine the size and operating parameters required for such a machine. In other words, the Bureau asked the question, "What kind of crusher would do the most good for the greatest number of mines?"

The Rapidex study, which involved contacts with 44 mines, established that a large potential market exists for a machine with these parameters:

Headroom	8-12 ft
Width	8-10 ft
Feed size	30-36 in.
Product size	Minus 6-8 in.
Throughput	100-300 t/hr
Moveup time	One shift downtime

The study also recommended a "pressure-type" crusher (i.e., jaw or gyratory) to meet the requirement for long-term, reliable operation while crushing the hardest materials (+30,000 lb/in.^{2}).

MODEL TESTS

Based on these parameters, the Bureau awarded parallel contracts to design, build, and test quarter-scale working models of two distinctly different crusher concepts, with the objective of determining the efficiency of each concept while minimizing the government's investment. The two contracts were awarded through a competitive procurement to Rapidex Inc. and Eagle Crusher Co.

Rapidex developed a "rotary-jaw" crusher, consisting of a fixed outer jaw and an eccentric shaft.[4] The name derives from the similarity of the crushing action of the rotary-jaw crusher to that of a conventional jaw crusher. However, the crushing chamber of the rotary jaw crusher has a curved geometry that permits horizontal feeding—an important consideration for a crusher that must operate in the restricted headroom of an underground mine. A unique feature of the design is a ring bar grizzly that bridges the interface between the feeder and the crushing chamber, while dropping the fines directly onto the discharge belt.

Eagle Crusher Co. offered a "tilted-jaw" crusher that also incorporates unique features.[5] Instead of the eccentric-pitman-toggle arrangement of conventional jaw crushers, the tilted version transmits the eccentric motion through movable side plates to the opposite, or lower, jaw member. The motion imparted to the movable jaw provides it with a positive feeding action, which compensates for the decreased gravity action of the tilted geometry.

The quarter-scale models of the rotary-jaw and the tilted-jaw crushers were subjected to a series of rigorous tests to measure throughput of materials having a wide range of hardness. The measured throughput was then extrapolated to estimate the production of the full-size versions of each design. In both cases, the model performance fell within the limits previously established, and both versions were deemed technically acceptable.

FULL-SCALE TILTED-JAW UNIT

The two contractors then submitted cost proposals for the erection and testing of full-scale versions of their respective designs, and a contract was awarded to Eagle Crusher Co., Galion, Ohio. The patented tilted-jaw concept was designated the Miller model 44 jaw crusher, and the jaw opening for the underground version was established at 36 x 40 in. The total system consists of an apron feeder, vibrating grizzly, tilted-jaw crusher, and discharge conveyor. Following assembly, the full-scale crusher system underwent acceptance tests at the contractor's plant. A variety of boulders were crushed, the hardest being traprock with a compressive strength of about 32,000 lb/in.^{2} These tests identified some problem areas and led to some modifications, the most important of which was the substitution of the apron feeder for the original chain feeder, which proved too fragile for the heavy duties imposed.

The research contract required that the system be tested in a surface mine or quarry by crushing 10,000 tons of hard material. The purpose of the extended testing was to identify any needed additional modifications and to confirm that the crusher system was ready for underground testing.

To satisfy this requirement, the crusher and ancillary equipment were transported to the McCarthy quarry in Upper Sandusky, Ohio, where a hard limestone of about 30,000 lb/in.^{2} compressive strength is extracted. The crusher successfully reduced the required 10,000 tons to minus 8 in. at throughputs of 200-300 st/hr. Occasionally, large slabs bridged between the grizzly side plates, confirming the need for an oversize breaker, as mentioned below. In a few instances, spilled material accumulated under the discharge belt, necessitating shutdown for cleanup.

FUTURE DEVELOPMENTS

Underground testing of the tilted-jaw crusher is planned for 1982. The test will be on a production basis and is expected to last for about one year. Detailed maintenance and cost data will also be obtained. The mobility of the system is of particular interest, because rapid moveups to keep the crusher close to the mine face will be of critical importance.

In a parallel effort, the Bureau is developing an oversize breaker through another research contract. The unit is primarily intended to prepare feed for the crusher by reducing oversize boulders, which result frequently from blasting and can be detrimental to the smooth flow of material. Prototype testing for the oversize breaker is expected to be done at about the time underground testing is completed for the crushing system. With the combination of breaker and crusher, the Bureau intends to make available to industry a complete underground size-reduction system.

Although the rotary-jaw crusher was not developed beyond the model stage, the author feels that it is still viable and potentially suitable for either surface or underground installation. The crusher is covered by a government-owned patent available for licensing, and interested persons may contact the Bureau of Mines for further details. ■

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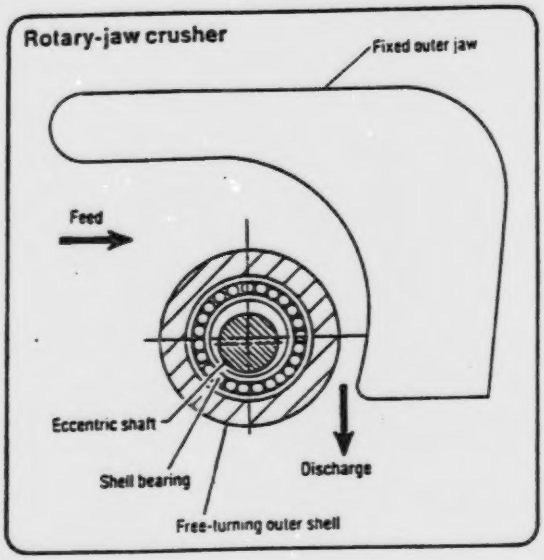
3) Rapidex Inc., "Application study of portable underground hardrock crushers, final report," USBM contract J0265006, December 1976, 123 pp.
4) Rapidex Inc., "Portable low-profile crusher for underground mining applications: Phase I—Design," USBM contract J0275043, November 1978, 81 pp.
5) Eagle Crusher Co. Inc., "Portable crusher for underground mining applications and design," USBM contract J0285003, Sept. 1978, 104 pp.

Rotary-jaw crusher



The quarter-scale working model of the rotary-jaw crusher concept may provide a basis for future development work.

FEEDER-BREAKERS AT WHITE PINE COPPER

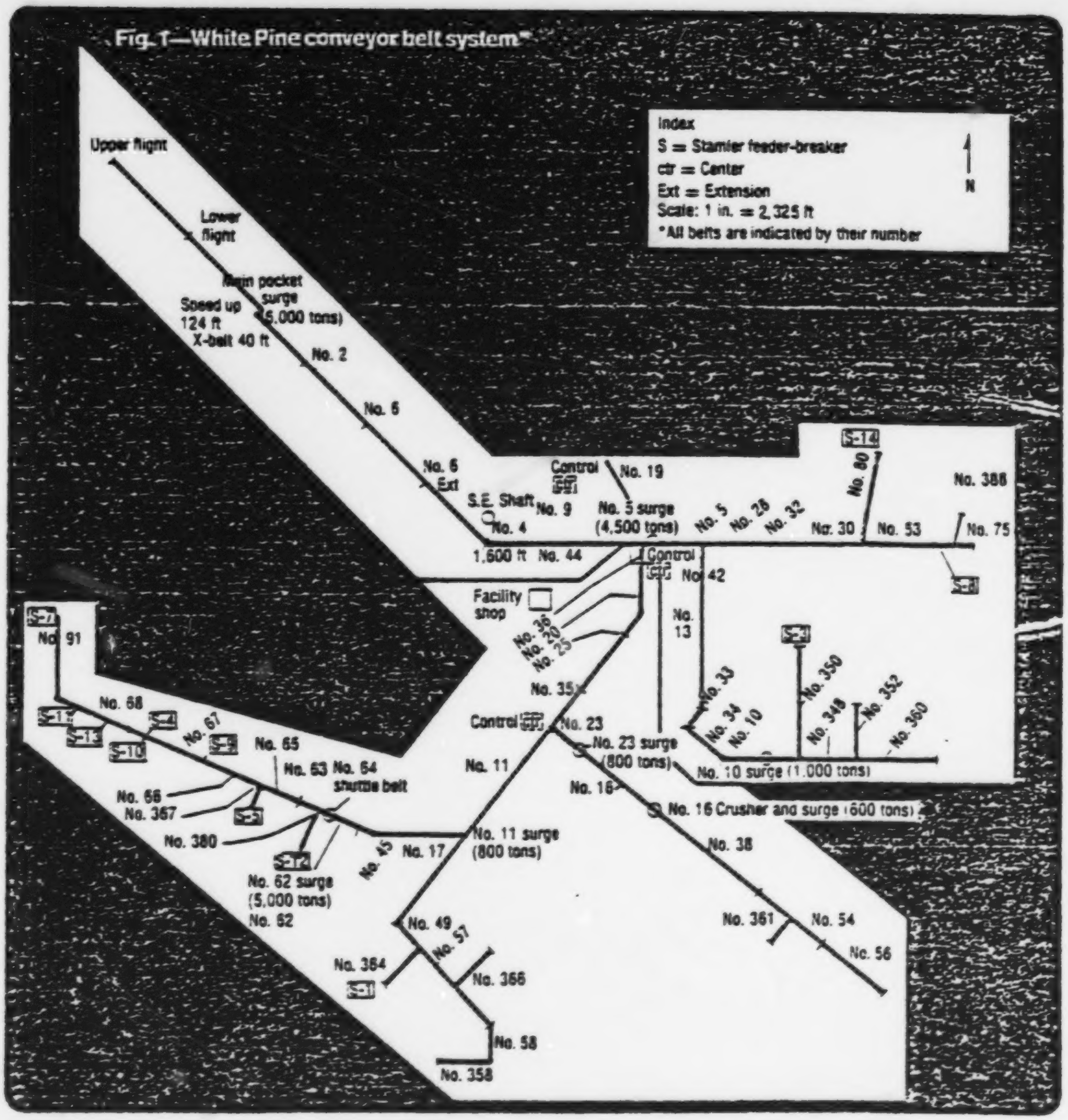


Fig. 1—White Pine conveyor belt system*

Index
S = Stamler feeder-breaker
ctr = Center
Ext = Extension
Scale: 1 in. = 2,325 ft
*All belts are indicated by their number
N
Upper flight
Lower flight
Main pocket surge (5,000 tons)
Speed up 124 ft
X-belt 40 ft
No. 2
No. 6
No. 6 Ext
Control
S.E. Shaft
No. 9
No. 4
No. 19
No. 5 surge (4,500 tons)
No. 5
No. 28
No. 32
No. 30
No. 53
No. 75
No. 388
No. 80
S-14
S-8
1,600 ft
No. 44
Control
No. 42
No. 13
Facility shop
No. 36
No. 20
No. 25
No. 35
S-3
No. 33
No. 34
No. 10
No. 350
No. 348
No. 352
No. 360
Control
No. 23
No. 23 surge (800 tons)
No. 10 surge (1,000 tons)
No. 16
No. 16 Crusher and surge (600 tons)
No. 38
No. 361
No. 54
No. 56
S-7
No. 91
No. 68
S-11
S-13
S-4
S-10
No. 67
S-9
No. 65
No. 63
No. 64 shuttle belt
No. 66
No. 367
S-5
No. 380
S-12
No. 62 surge (5,000 tons)
No. 62
No. 45
No. 17
No. 11
No. 11 surge (800 tons)
No. 49
No. 57
No. 384
S-1
No. 366
No. 58
No. 358

A NEW CONCEPT IN HARD-ROCK MINING LOWERS ORE HANDLING COSTS

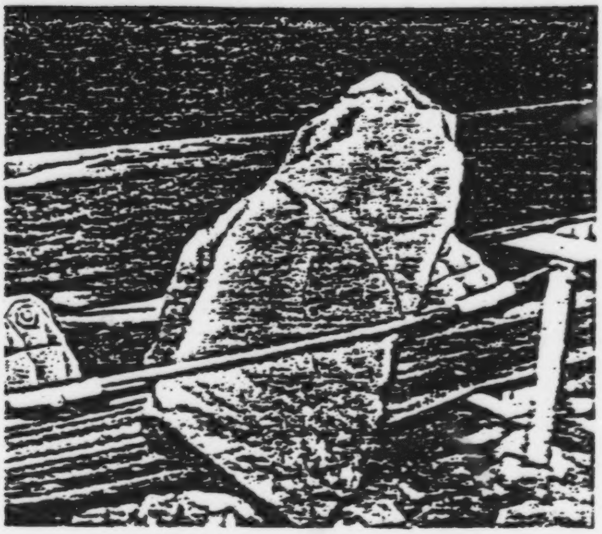


Fig. 2—Oversize ore blocks caused severe edge damage and spillage.

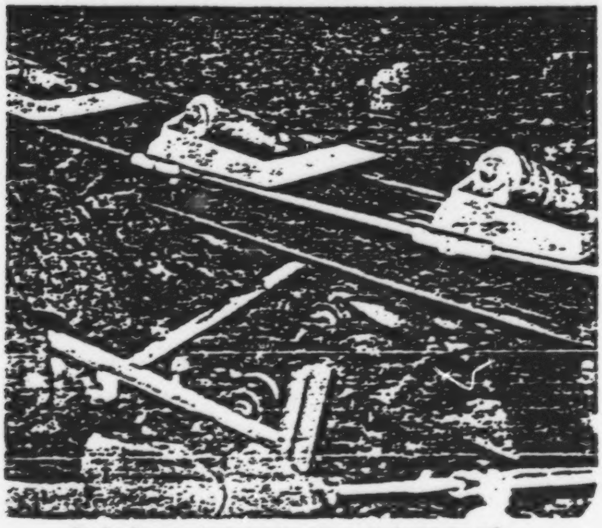


Fig. 3—Oversize chunks knocked down a stand, causing a spill.

Robert E. Niemela, director of mine planning and engineering
White Pine Copper Div., Copper Range Co.

The White Pine Copper Div. mine uses load-haul-dump (LHD) vehicles and ore cars to haul ore to low-profile feeder-breaker units, which crush the ore and feed it to an extensive underground conveyor system (Fig. 1). Prior to converting to feeder-breaker units, uncrushed ore was fed directly to conveyor belts, which carried the ore to centralized crushing stations. Significant delays, a common occurrence in this system, resulted in low conveyor capacity and availability and lead to the introduction of the feeder-breakers.

The White Pine Copper Div. of Copper Range Co. is located in Ontonagon County in Michigan's Upper Peninsula, 6 mi south of Lake Superior. The White Pine orebody lies in the bottom beds of the Nonesuch formation, a 600-ft-thick series of siltstones, sandstones, and shales. For the most part, economic copper mineralization occurs only in the lower 20 to 25 ft of the Nonesuch formation. Chalcocite is the primary mineralization, with native copper and silver also present. Mining height, which varies from 7.5 ft to 14 ft, is determined by grade. Mining depth has varied from 40 to 2,200 ft. Approximately 9 sq mi of the orebody has been mined by trackless room and pillar methods during 26 years of mining.

A LOOK AT PAST PROBLEMS

Prior to 1970, gathering arm loaders were used in the mine's production headings to load ore into shuttle cars. Loaders were replaced in the early 1970s with rubber-tired LHDs. Two to four shuttle cars per LHD, depending on haulage distance, moved the ore to loading pockets, which incorporated vibrating feeders. These feeders discharged the run-of-mine ore onto 48-in. conveyors, which in turn carried the ore to centralized crushing stations. Grizzly shaker feeders at these stations scalped the minus 6-in. product and fed the oversized ore to either 21 x 42-in. or 32 x 40-in. jaw crushers. The discharge from these centralized crushing stations was transported to the surface by a combination of 48-in. and 54-in. conveyors.

A typical problem in many underground mines is the inability to efficiently utilize manpower and major pieces of equipment during ore handling, due mainly to the uneven capacities of different segments of the ore handling operation. This imbalance became painfully apparent when management considered increasing the rate of copper production in the late 1960s. The conveying system was viewed as a limiting factor in the consideration of any future

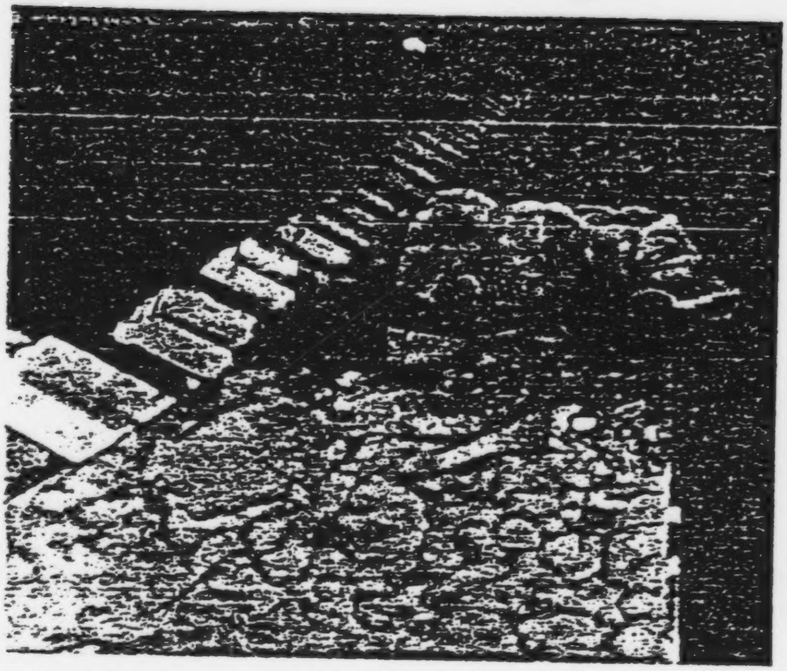


Fig. 4—Oversize ore caused this belt to run off, resulting in spill.

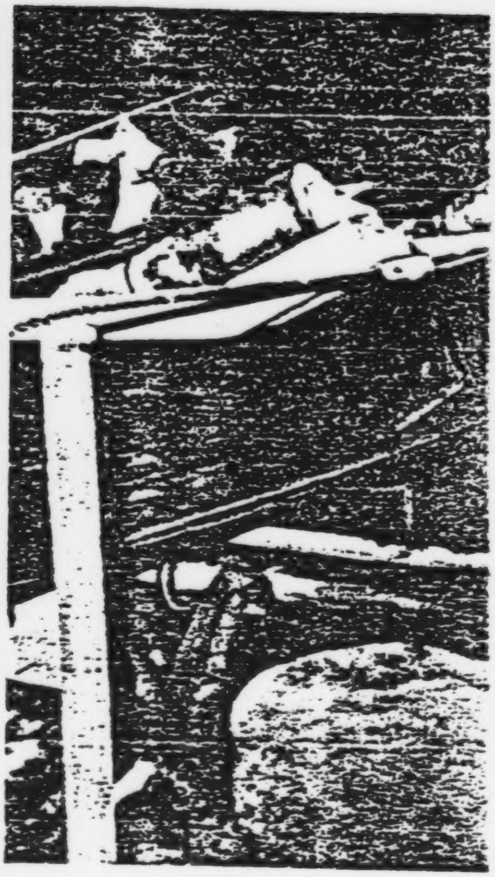


Fig. 5—A spill was caused when this stand was knocked down by oversize ore.

tonnage increases. The system was actually overtaxed at the then existing 16,000-tpd level even though the rated capacity was above this limit. Decreased conveying capacity was due mainly to outages caused by spillage or damaged components due to oversized ore and slug loading. Problems created by handling oversized rock were common (Figs. 2-5).

Low belt availability, excessive manning requirements for the conveying system, and the potential for accidents due to handling oversized ore influenced management to contract consulting services for a definitive study of the White Pine materials handling system.

RESULTS OF THE STUDY

The study found that the existing conveying system had adequate conveying capacity for existing production. Also, handling of oversize material and inadequacies at transfers and crusher stations caused restrictions in material flow. These restrictions created an irregular operating procedure with a resultant loss of production potential and excessive operating personnel requirements.

Also, the study determined that production objectives established for the late 1960s and early 1970s would be impossible to achieve without restrictions on feed size to the existing system, design changes to existing installations, additional crushing equipment, and precise control of conveyor feeding and operation. The study also noted that productivity was negatively affected by the unbalanced capacity of the loading pockets and haulage equipment, and oversize from the jaw crushers caused reduced mill capacity at the other end of the system.

The following recommendations were made:

- Improve flow at transfer stations.
- Restrict the top size of uncrushed feed to the system to a maximum of 2 cu ft.
- Restrict the feed rate to feeder conveyors to 500 tph.
- Employ a control system to shuttle one car load at a time into the mainline conveyors.
- Crush all material to minus 8 in. before it is discharged onto mainline conveyors.

In a followup study, the White Pine plant engineering department detailed mill outages due to oversize from underground crusher stations. The recommendations of this report dealt with modifying drill rounds and/or underground crushers to eliminate slabs and thus increase mill availability.

An additional report published in early 1973 stated that "top priority should be given to reducing the ore at or near the face, before it enters into any belt line." More recently, in 1974, a study was completed under a US Bureau of Mines contract (No. H0230004—Theodora Barry and Associates, July 12, 1974) on ore handling systems in general. An excerpt from this report:

> "Ore handling systems throughout the underground metal mining industry are adversely affected by the presence of oversize muck. One of the major findings of our detailied studies of five mines was that the oversize ore is the single most consistent denominator responsible for reducing the overall productivity of the

ore handling system. Additionally, it was found that the 'oversize muck' was responsible for the creation of a number of hazardous situations and operations underground. Operations like secondary blasting, blasting large boulders, clearing chute hangups, cleaning ore spillage, etc. can often be attributed to the presence of oversize rock. The result in each case is reduced operator productivity and safety.

We recommend that the US Bureau of Mines undertake a research program based on new technology or to modify existing technology to develop a high-energy impact rock breaker. The benefits to the mining industry as a whole, from increased productivity and operator safety, will be significant."

A recent USBM survey of five underground mines showed that coping with oversize rock continues to be a major source of safety-, energy-, and economic-related problems in the mining industry. Notable in the study was the universal occurrence of the oversize problem among very different orebodies and mining methods. Surprisingly, uniform dimensions of the oversize ore from very different mining operations was roughly 1 x 2 x 3 or 4 ft, and the proportion of oversize material was about the same, approximately one-fifth, regardless of the orebody and mining method. One can be certain that the proportion of accidents, inefficiencies, and wasted expenditures caused by this oversize is disproportionately higher.

It was apparent that some system for ratio feeding the ore to the conveying system was necessary. The ore would have to be crushed as early in the cycle as possible. The need for ratio-fed and crushed ore at or near the production face, plus advances made in feeder-breaker design in the coal industry, prompted White Pine to investigate feeder-breaker applications for the White Pine shales. The shales have a compressive strength of 12,000 to 28,000 psi.

SELECTING THE EQUIPMENT

One of the feeder-breakers chosen for testing in 1973 was manufactured by Stamler. This unit, model BF-14-8-10, incorporates a flight chain conveyor to move run-of-mine ore from the 20-ton hopper through a sizing region and onto a feeder belt at a given rate. Size reduction is accomplished by carbide-tipped bits mounted on a rotating breaker head. The breaker is installed with 6 in. clearance above the flight conveyor deck. The compressive penetration of the sharp picks into the rock and the impact cause the rock to fail quickly in tension due to wedge action.

Three Stamler feeder-breakers were purchased and installed for production testing in upper shale mining units. A sharp increase in mining productivity was observed as each machine was put into service. The need to station personnel at conveyor transfers also was eliminated as each pocket area was converted to a total crushed-ore, ratio-feed system. This trend toward higher face productivity and reduced personnel requirements lead to the purchase of eight additional machines to crush all mine shales. Total mine conversion to feeder-breakers was delayed until more operating and cost data could be documented with regard to their application in the highly abrasive (60% silica) sandstone formations.

Experiments indicated that an approximate 40% increase in crushing cost in this hard-rock application was more than offset by increased face advance in the development area. The weak copper market in early 1979 brought on fluctuating production levels that made necessary rapid changes in production areas. The feeder-breaker system was ideal to provide the necessary flexibility. Currently, 30% of all mine rock is crushed with Stamler feeder-breakers.

Six years experience with feeder-breakers at White Pine has resulted in several modifications to both the machines and associated installations. The original concept of the feeder-breaker at White Pine was that of a self-propelled, highly mobile crushing machine that required no excavation. This was the image created by the use of feeder-breakers in underground coal operations. Application at White Pine has deviated from this concept. The size and weight (42 tons) of the machine, dependent upon the attitude of roadways and belt drifts, are not amenable to crawler-type applications. Machines are provided with a wheeled towing arrangement incorporating an independent diesel-powered prime mover, which has made the machine highly mobile. Steep grades underground, the size of the conveyor equipment, and the geometry of the dump pocket arrangement have made it impossible to install feeder-breakers without excavation.

Fig. 6 shows a typical feeder-breaker installation. Excavation height is approximately 15.5 ft, which is considerably less than the approximately 25-ft minimum requirement for conventional jaw crushing installations.

White Pine experience has demonstrated that, depending on the rock type, feeder-breaker operating cost (picks, maintenance, etc.) is considerably more than that of a conventional crushing, feeding system. However, this increased cost is offset in a number of ways:

- A reduced pillar height, the relatively short feeder-breaker installation period, and the reduced roof and pillar damage due to blasting (less than one-tenth the explosive requirements for excavation) have virtually eliminated supplementa-

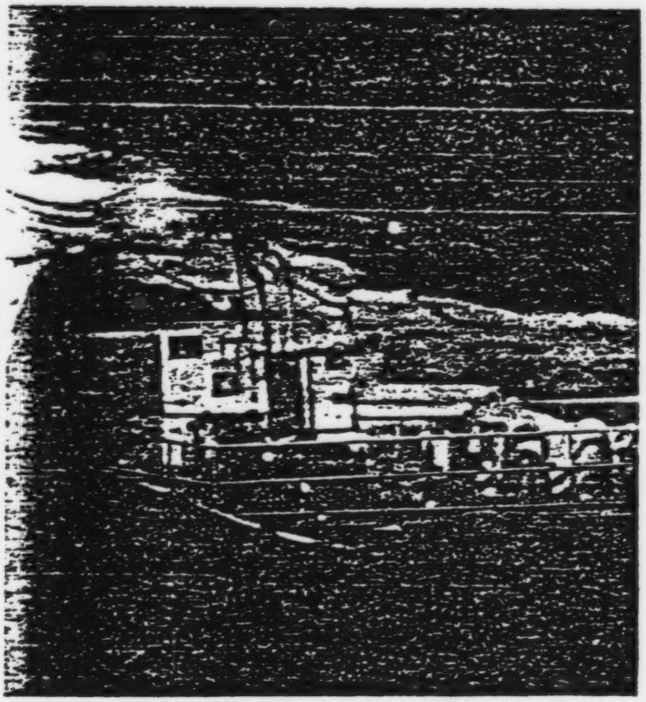


Fig. 6—A typical Stamler feeder-breaker installed on a 36-in. feeder belt.

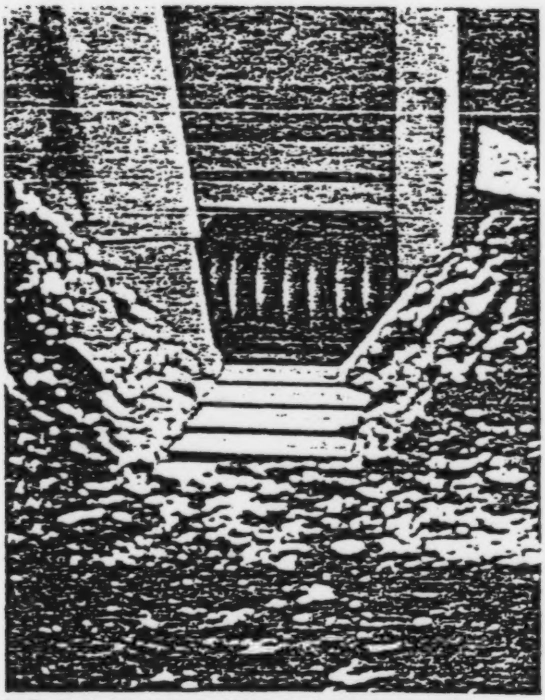


Fig. 7—A typical dump approach for ore at a feeder-breaker installation.

ry ground support. Additional support for a typical crusher installation costs $20,000 to more than $100,000 over the life of the installation.

- A typical crusher-feeder installation will cost approximately four times that of a feeder-breaker installation. This is due to excavation and massive structural work for the crusher alone plus excavation requirements and structural work for ramps and feeders. The bulk of a feeder-breaker's support structure is not only standard in all installations but also reusable.
- The additional six to eight weeks required to install a crusher affected the belt construction schedule and, consequently, the belt support system and haulage. It was desirable under the old system to advance truck dump pockets at six-month intervals. More often than not, it was physically impossible to achieve this target because of the work load on construction. The result was long hauls for up to a year before dump pocket relocation. The simplicity and speed of the feeder-breaker installation and the ability to install the machine at intermediate points on a conveyor flight have also allowed for short lead time changes in the belt support plan. The result has been shorter hauls to dump pockets, higher productivity, and greater production flexibility.
- The relatively higher maintenance costs of the feeder-breakers are further offset by increased belt and component life. Belting, the most expensive part of the conveying system, is being protected from abrasion, impact, and runoff, due to the minus 10-in. material and ratio feeding. The first belt salvaged from an obsolete ratio-fed ore line went into immediate use on a newly constructed conveyor in another mining area. The belt was in excellent condition and service life is estimated to be at least double that of belting used on the old system. Similar increases in idler life are also expected.
- Ratio-fed, crushed ore has allowed conveyors to be suspended from the back, showing promise of cheaper installation and significant reductions in conveyor maintenance, cleanup, and salvage costs.
- Significant increases in belt availability, currently 95%, has accounted for a 10% to 15% reduction in underground stockpiling.
- Elimination of slug loading with limited ratio feed has accounted for increased conveyor capacity along mainlines. The plan now includes more dump points feeding to the mainline. More ore is consequently transported from an area in a given time.
- Size reduction along with ratio feed and elimination of slug loading accounted for a 50% reduction in personnel required for conveyors. This drop in manpower was effected by eliminating the need for transfer operators.
- As the belt system was converted to crushed ore, the accidents due to oversize dropped proportionately.
- Sharp increases in productivity were noted after converting from truck to LHD haulage.
- Road maintenance requirements dropped by as much as 75% in those areas where conversion to LHD systems occurred.
- Mining heights where LHDs were incorporated were lowered, and corresponding benefits in grade were realized.
- As feeder-breakers were introduced into the system, mill throughput rose noticeably due to an increase in minus ¾-mesh material and the reduction of oversize.

All of these benefits have helped to offset the high primary crushing costs associated with the feeder-breaker system. These costs will be reduced with further experience.

The feeder-breaker is still a relatively new concept in hard-rock applications. Changes in wear material, components, and breaking technology already have been made. Each improvement has made application of the machine more viable. Additional future experience promises to lower system costs. ■

High Speed Belt Conveying

by
P D H Comley *CEng MIMechE*

Troughed belt conveyors have been with us since the turn of the century. They have evolved from relatively crude devices, carrying small quantities of loose bulk material over short distances, to today's complex and highly sophisticated systems, capable of transporting over many miles, loads of many thousands of tonnes per hour.

There are very few industries in the world which do not involve the handling of loose bulk materials, at some stage of the manufacturing cycle, and these invariably require conveyors to be an integral part of the system.

There are other means of transporting these materials, but these can involve the employment of a large number of personnel, many vehicles, and a complex back-up maintenance system.

The most common of these other transport systems is the dump truck. However, to employ the dump truck a suitable road system must exist, or else this must be constructed, a most expensive item.

Every dump truck requires a driver, and for any reasonable tonnage to be conveyed, several dump trucks will be needed. The greater the tonnage, allied with increasing distance, the size of the dump truck fleet increases in direct proportion. Remember also that every truck spends 50 per cent of its operating life returning empty, not taking into account the time for filling.

If a two or three shift system is worked, then the number of drivers involved doubles or trebles. In addition to this, spare vehicles must be held to replace those that are in for overhaul or which have broken down.

Where does the overhaul and repair take place? A complete garage facility is necessary, with skilled mechanics, the necessary specialist equipment, and administrative staff to control the whole exercise.

The dump truck is also a most unsociable animal. No one is enthusiastic to have noisy, dusty vehicles traversing inhabited areas, with all the disturbance that this brings. As the world demands more and more output of our natural resources, then the sources near to inhabited areas require to be exploited, as well as those in the remoter regions or those in areas of natural beauty.

In all these cases vehicular transportation creates a demand for more manpower, causes more environmental problems, and is subject to loss of output that ensues from disruption caused by extreme weather conditions.

Alternative means such as rail transport can be reasonably viable if the existing rail system can be utilised. However, for many applications track would require to be specially laid (an extremely expensive operation), and trains, while carrying a much larger tonnage than dump trucks, still involve a considerable investment in rolling stock. The amount of rolling stock required must take account of the same problem as with dump trucks, of operating empty for 50 per cent of their working life.

Maintenance of track, rolling stock and engines is high, although the replacement requirements are considerably less than with dump trucks.

Where the terrain over which the material is to be transported is of a difficult nature, then a road or rail track will require careful grading, bridges of a suitable design to handle the heavy loads involved, and proper drainage.

These are just some of the major aspects that must be considered where loose bulk material is to be transported from point 'A' to point 'B'.

Consider now the conveyor, or conveyor system. It requires no special road to be constructed, just a grading of the land, and the conveyor stools mounted on simple sleepers or supports.

Steep gradients, both uphill and downhill can be easily scaled, and where bridges are required these can be simple in construction, since the loads are relatively light, and are uniformly distributed.

A conveyor can work up to 24 hours a day, with no interruption of the flow of material, and can be virtually immune to weather conditions by the use of suitable covers.

As far as the environment is concerned, a conveyor can be made to blend into the surrounding landscape by the use of covers and suitable painting. It creates very little noise, and can be engineered to avoid dust problems.

In terms of maintenance, this is largely a matter of replacing the occasional idler roller, and ensuring that the belt remains properly tracked. The life expectancy of today's major conveyors is extremely long.

As for personnel requirements, the conveyor demands the minimum of staff, an operator (who may well be in overall charge of a complete conveyor complex), and one or more maintenance men, who would normally be employed in patrolling the system to ensure correct functioning of all items, thus reducing their maintenance role to a minimum.

Finally, a conveyor is powered by electricity, and is therefore a greater conserver of our depleted stocks of fossil fuels, when compared to dump trucks and engines that need diesel fuel to function.

Having, I hope, shown that the troughed belt conveyor is a vital and an increasing requirement, in the field of bulk material transportation, the remainder of this Paper is devoted to the aspect of high speed conveying.

WHY DO WE NEED HIGH SPEED?

Basically to optimise output, and keep conveyor width to a practicable minimum.

Our first consideration must be the size of material to be carried, as this may well dictate the minimum width that is acceptable. The elimination of spillage is of the utmost importance, therefore there must be adequate edge clearance.

The conveyor must be capable of handling the maximum output that will be fed onto it, bearing in mind the nature of that material, and its natural surcharge angle.

If a conveyor is run at slow speed, then a wide belt is necessary, the conveyor supporting structure, troughing and return idlers must all match this width, as well as head end units, receiving sections, etc.

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The weight of material per unit run is high, therefore the supporting idler sets have to be closely pitched, and the number of supporting stools increases in direct proportion.

All of these factors add considerably to the initial capital expenditure, and the cost of replacement parts is proportionately high. Space may be at a premium if the conveyor is to be located in a tunnel or roadway, or on a jetty, or any other location where access past the conveyor is necessary for the movement of men and/or equipment.

By specifying a high speed conveyor, width is reduced, and the capital outlay is minimised. Idler pitching can be increased, spares holding reduced, and the minimum necessary space utilised.

The maximum speed chosen must take due account of the parameters specified at the beginning of this section, and care must be taken to ensure that a dust hazard is not created.

You may well say "that is all very well, but what about the power costs, higher speeds mean higher power consumption". True to a point, but consider that the wider, slower belt has greater weight of moving parts, has more idler sets per unit length, and therefore may cost just as much to operate at low speed as the lighter, narrower high speed version.

If the terrain to be traversed is uphill, then a major proportion of the power consumption is in moving and lifting the material. Therefore, a higher speed will be even more cost effective than that for a conveyor which is located on a relatively level gradient.

I said at the beginning of this section that we need high speed to optimise output. This is partly true. If the feed onto the conveyor can be maintained at a consistently high level, then all the factors discussed are relevant.

However, it is more frequently the case that factors dictate that the feed rate will vary from shift to shift, and even during a shift. This is where we come back to optimising the output.

For such applications there is considerable merit in choosing the highest practicable speed for the maximum feed condition, and reducing the speed as the feed rate reduces. In this way we keep operating costs to a minimum, reduce unnecessary wear and tear on all moving parts, and, in short, optimise the output.

We therefore come to what I consider to be one of the most important developments that have occurred in recent years in troughed belt conveying, the variable speed conveyor. In the following sections of this Paper are discussed the major factors and design considerations that must be taken into account when specifying such conveyors.

MAJOR FACTORS TO BE CONSIDERED

Reliability

This is probably the most important factor that has to be considered. There are very few operations today, if any, that are not going to involve a serious financial loss, either directly or indirectly, due to a lack of reliability, and a consequent breakdown of the equipment.

Consider what happens in a bulk material handling operation if a breakdown occurs:

(i) The feed system has to close down
(ii) The source of the feed has to be discontinued. This may mean the stopping of the unloading of a ship, the operating of a coal face, or the discontinuation of output at a preparation plant. Bunkering facilities may make it possible to cushion the effect, but this is expensive, and may only be partially effective
(iii) The output or completion of the final product of a plant may be seriously disrupted, causing a breakdown in delivery schedules
(iv) Workers are temporarily laid-off, or put on to short-time, but still have to be paid.

As an example, a colliery with two faces producing jointly 500 tons/h of coal, would sustain a loss of at least £13000 to £14000 every hour through loss of output, in the event of the main trunk conveyor breaking down. A breakdown lasting only one day, could well equal the initial outlay in purchasing that conveyor.

What do we mean by reliability? First, the fewer the units the less likely there is to be a breakdown.

If one conveyor will do the job, why specify two or more. The reliability factor is increased in direct proportion.

A belt conveyor is by nature a simple machine. This characteristic goes a long way towards making it reliable, and its reliability is further ensured if its major component parts are of robust and proven design, and care is taken to preserve the quality of simplicity by giving detailed consideration to each item of the equipment:

Drive arrangement that keeps the accelerating torque as low as possible, and extends the accelerating time to the maximum, thus minimising the starting tension.

Where appropriate, using a variable speed drive so that the conveyor runs along at that speed necessary for the particular loading. This permits a saving in energy, a reduction in wear and tear, and increases the operating lives of all components.

Tensioning arrangements that automatically adjust to match the various load circumstances, running and starting on the one hand, and decelerating (braking) on the other, and which maintain tension in the belt when the conveyor is stopped, thus avoiding undue belt sag and consequent increase in belt tension on re-start.

Conveyor structure and idlers that are adapted to the load and speed, and in which both troughing and return sets can be accurately aligned. The more perfectly a belt can be aligned to run true, the greater the reduction in belt scrubbing due to off-centre running and loading, the elimination of spillage, and the consequent reduction in wear and tear, and lower expenditure of energy.

Receiving points designed to transfer the product onto the main belt in such a manner as to minimise its fall, centralise it, and to pre-accelerate it, thus ensuring proper protection of the belt.

Cleaning equipment located at the drive end to eliminate dirt build-up on pulleys and return belt idlers.

A conveyor belt that is chosen for its suitability for the application, to have a safety factor that is adequate for the duty, and has the minimum number of joints, which must be made to the highest standard.

Finally, reliability is the relationship between the actual operating time and the time for which the operator could have used the installation. In a conveyor system operational reliability is inversely proportional to the number of conveyors ensuring a link-up, to the number of parts constituting a conveyor, and to the quality and suitability of the constituent parts.

In all cases the aim must be to provide the highest 'availability coefficient' possible.

Optimum Life Ratings

I have discussed at length, in the previous section, the need for reliability, and this must now be related to the planned operating life of the equipment.

Consideration must be given to the cost of replacement, and the shut-down time involved in the replacement.

From these considerations the optimum life ratings of the constituent parts can be established. Account must be taken of the tonnage to be carried, the number of operating hours per annum, and the overall life requirement.

Generally speaking, the following parts of a conveyor can be considered in ascending order of life expectancy:

Idler rollers: these should normally have a life expectancy of 5 to 10 years, and are generally easily replaceable items

Pulleys and main bearings: items that require at least a day or more shut-down to replace, and should therefore have a life expectancy of 7 to 10 years

Conveyor belt: while this is almost invariably the most expensive part, its life is largely determined by wear.

Therefore type of covers and joints must have special consideration. For solid woven or ply belts life expectancies of up to 8 years can be considered realistic. For steel cord belts lives of 15 to 20 years are desirable. Replacement is expensive and time consuming

Drive units and gearboxes: these should last the life of the conveyor, but may be considered on a life rating of 7 to 10 years or more, on the replaceable items such as seals, bearings, etc

Conveyor structure: this should last the life of the conveyor.

Where components are subject to high wear, and fairly frequent replacement, such as skirt plates, chute linings, skirt rubbers and belt scrapers, ratings are not normally considered, but special consideration must be given to ease of maintenance and replacement.

Maintenance Requirements

I have already drawn attention to the necessity of designing for maximum reliability, and minimising the number of constituent parts of the conveyor. This of course helps to reduce maintenance, but does not preclude this most important aspect.

For maximum reliability, reduction in wear and tear, cleanliness of the equipment, planned maintenance is essential.

Care must be taken to ensure that all bearings are properly lubricated, and re-lubricated at the specified intervals. This also applies to the maintenance of oil levels in gearboxes, fluid couplings, hydraulic power packs, etc, and replenishment of the oils with the correct grade of lubricant, at the specified intervals.

The maintenance schedule must draw particular attention to the need for cleanliness of the system, and proper alignment of pulleys and idler sets.

The design of the conveyor will have included means for cleaning the belt, which will be of the mechanical type, but on special applications may involve the inversion of the return belt, or washing of the return belt.

These cleaning devices must be maintained and replaced on a very regular basis. Failure to carry out this maintenance will result in dirt build-up on pulleys and return idler sets. The consequence of this may well be extremely costly. The build-up of dirt on both pulleys and idlers will cause off-tracking of the belt, and extensive belt edge damage may result. In the case of steel cord belts, a build-up of dirt on bend pulleys can result in the fracture of the steel cords, at the point of build-up, and this will then not only be expensive in terms of repairs, but more particularly in loss of production while these repairs are being performed.

A further effect of dirt being carried back onto the return idlers, is that the dirt may build-up around the ends of the roller supporting brackets, causing friction, and an unnecessary increase in power requirements. This dirt build-up around the ends may also cause premature failure of the idlers due to forced ingress at the seal elements.

Misalignment of idlers causing off-tracking of the belt, will create additional scrubbing between the belt and these idlers, leading to wear on these components, as well as an increase in power consumption.

The correct tensioning of the conveyor is vital, as a reduction in tension will lead to belt slip at the drive. The consequence of this will be severe cover wear, leading to premature replacement of the belt, overheating at the pulley which might constitute a fire hazard in certain operations. In addition, there will be a build-up of the material on the belt, due to the speed reduction, and this could lead to increased slippage and a compounding of the problem.

A loss of tension at the tail end of the conveyor will increase belt sag, which may cause spillage, but will most definitely cause an increased resistance in the running of the belt, leading to increased power consumption, and overload on conveyor start up.

It is therefore of the utmost importance that the belt tensioning arrangements are always properly maintained, and kept in correct alignment, and are free running.

Maintenance must therefore figure highly in the concept of the conveyor. All parts requiring maintenance must be designed to ensure that the necessary maintenance can be carried out quickly and easily, and wearing or replaceable items must be capable of being quickly detached, replaced and aligned.

If an item requiring maintenance is difficult to reach, and/or involves the removal of other equipment in order to carry out routine maintenance, then human nature being what it is, the necessary maintenance will not take place, and a breakdown will result.

In short therefore, ease of maintenance must be designed into the conveyor, a planned maintenance schedule will be required, and consideration given to the need for any special maintenance equipment. This latter equipment could involve audio devices for listening to rotating equipment, such as idlers, to detect the start of failure, and/or equipment to aid the removal and replacement of idlers, possibly under operating conditions. All of these aspects must be taken into account.

Protection

A conveyor, or conveyor system, represents an investment, and that investment must be protected. I have already shown that reliability is a major requirement, but even with the most careful design, malfunctioning can occur, and parts will deteriorate due to wear and tear.

It is vitally important to provide warning of imminent failure or malfunctioning, so that corrective action can be taken before an actual failure occurs.

The warning should be arranged to operate with a suitable safety margin, and if the warning is ignored then after a pre-set time an automatic shut down must take place.

Failure to provide protection equipment will ultimately lead to a breakdown of the conveyor with damage being sustained. When such a situation arises, not only will the damage be expensive to put right, but the consequent loss of production may well be even more serious and expensive.

All operating parts must be studied very carefully, and consideration given to any malfunctioning that could lead to a breakdown and/or damage being sustained. Suitable protection equipment must be provided, and a condition display panel be located in the operator's work area. In a major installation this should be a proper control room with full display panels that incorporate identification of the type of fault, its whereabouts and either flashing lights or an alarm to draw the operator's attention to the occurrence of a fault.

In addition to the protection equipment it may also be desirable to have television monitors located at suitable points such as feeds and transfers, so that the operator can observe operating conditions at a glance.

While it is of the utmost importance to ensure that proper protection of the conveyor is provided, it is also equally important to provide protection for personnel who have occasion to be in the vicinity of the conveyor.

This must take full account of properly designed and adequate guarding of all parts that constitute a hazard. There are a number of safety standards which have been compiled by people having a detailed knowledge of dangers which exist, and the requirements to provide adequate safeguards, and these must be observed to the letter.

The Health and Safety at Work Act puts the onus on the designer and manufacturer to provide the necessary safeguards, and these must be an integral part of the design, and not something which has been added on as an afterthought.

Where guarding has been provided in an area which requires routine maintenance, that maintenance should be capable of being carried out without disturbing this guarding. A case in point is the lubrication of bearings, and this can be done from a remote control grease point.

Where this is impracticable, and guarding has to be removed to carry out the maintenance operation, then means should be provided to automatically isolate the conveyor on removal of these guards.

There is also a need to be able to stop the conveyor at any point along its length, and when this is operated provision is needed to identify the area in which the stopping action has taken place. This may be of vital importance if someone has become trapped and injured, so that help can be directed with the minimum loss of time.

On major conveyors, or conveyor systems, consideration should also be given to providing communications with the control room along the full length of the system. This has a number of advantages. If someone has been injured and a trip-out has occurred, the operator can keep in touch with the rescuers, and provide additional facilities or help if these are needed.

In addition, if a warning or trip-out has occurred on one of the protection devices, the facility of telecommunications provides the means to keep the operator informed on the extent of the problem, the progress being made to rectify the situation, and to be able to call on additional help and materials as required. It also enables a two-way check to be made on the satisfactory completion of the repairs, and re-setting of the protection devices.

In high speed conveying all these factors take on an even greater importance, and very careful consideration must be given to adequate working clearances, condition monitoring, and guarding to the highest standards.

Economics

Having considered all aspects of reliability, optimum life ratings, maintenance requirements and protection, the overall economics of the conveyor must be carefully studied.

The capital outlay of the conveyor or conveyor system, must be the minimum that is commensurate with its operating at the most economic level, taking due account of running costs, operating staff and maintenance.

If the conveyor is to be operational for many years, then the operating costs, and the cost of replacement parts must be carefully evaluated. With inflation continuing to rise, these costs are going to increase considerably during the life of the installation, adding to the selling price of the final product.

Therefore, the belt speed, belt width and type of idlers to be used, must be considered carefully, to minimise the power requirements. This must take into account the use of high quality idlers, with the minimum resistance to turning, keeping belt sag as small as possible, and ensuring that alignment of all structure and idlers is as perfect as possible.

Wherever possible a single conveyor should be specified in preference to two or more. Not only does this reduce the maintenance requirements, it increases the life of the belt, generally the most expensive single item in the conveyor. The life of the belt is determined by a combination of stress cycles and wear. A belt is stressed every time it passes around a drive or bend pulley, and wear takes place predominantly at the loading points. The longer the conveyor, the less frequently any one part of the belt is subjected to a bending stress, or passes through the loading section.

I have made a strong point under reliability, of having the minimum number of component parts in the make-up of a conveyor, however, this must be assessed very carefully. It is not economical to have a single drum drive for a horizontal conveyor, as an unnecessarily heavy belt will be required, due to the large tensions imposed by this drive arrangement. In these circumstances a dual or multiple drive will require to be assessed, and the greatest economy on a very long conveyor may be achieved by specifying drives at both head and tail.

On an inclined conveyor, the slope weight of the belt will provide a considerable measure of the tension required to drive the belt, and above a certain incline a single drum drive will be quite sufficient, no belt tension advantage being gained by using dual or multiple drives.

Where multiple drives are chosen, economy of maintenance and spares holding will benefit by standardisation. As an example, if a 2 to 1 drive ratio were chosen, it is likely to be more economical to choose three identical drives (two on the primary and another on the secondary), instead of two drives of 2 to 1 power ratings.

There are many other detail areas that have to be taken into account, but I think I have drawn attention to the philosophy that has to be adopted, without making this paper unnecessarily long.

DESIGN CONSIDERATIONS

The Drive Unit

In this Paper, I am mainly concerned with examining the aspects of high speed conveyors, so the types of drives discussed do not cover the more basic ones commonly used on the majority of conveyor installations.

Having established the required belt speed, the power requirements, and whether a single or multiple drive configuration is to be employed, main consideration must be given to providing as soft and as smooth an accelerating force as possible.

A limited acceleration period can be provided by employing a fluid coupling. However, there is always a considerable amount of slip present in a fluid coupling, while it is accelerating to full speed, and this slip converts to heat which has to be dissipated by using coolers.

The design of such couplings, where the starting torque, under full load conditions, is to be kept to a low level, is fairly complex. Oil levels have to be well maintained, and in multiple drives it is difficult to achieve proper matching of each coupling, one to another.

The preferred arrangement is to use an electric motor that can provide the desired low acceleration curve, and can either be directly coupled to the drive drum, or through a suitable reduction gearbox.

There are several types of motor and control gear available, and some of these are:

Slip Ring Induction Motor, with Liquid Controller

This provides a soft, slow speed start-up of the conveyor, but is only capable of running, under loaded conditions, at full speed rating. However, if unloaded belt maintenance or belt inspection running is required, a slip power recovery system can be provided to permit continuous running of the unloaded belt at speeds between 1.5 per cent and 10 per cent of full speed.

Briefly, the acceleration is controlled by fixed and moving electrodes. The fixed electrodes are mounted on a rigid structure supported by top channels, and contained within insulating electrode chambers (one per phase).

The moving electrodes are mounted on a carriage supported on rollers by the overhead channels so that they can move horizontally towards or away from the fixed electrodes.

The electrolyte is pumped from the bottom of the unit into each electrode chamber. When the electrodes are far apart the level of the electrolyte is low and hence the resultant resistance at its greatest.

As the electrodes travel nearer, the level of electrolyte increases until it finally overflows the chamber and completely immerses the electrodes. The resistance at this point is very small.

With this arrangement a very high resistance gradient can be achieved, which is necessary to give the degree of precise control required.

In operation the moving electrode carriage is connected by a link mechanism to an hydraulic servo mechanism.

The servo is so designed that when the operator presses the start push button, the servo moves the electrodes forward to the point at which the current flowing is sufficient to start the motor. From there on the rate of acceleration is determined by a valve, controlled by an adjustable ramp function, such that the torque exerted is within the maximum allowed by the belt design.

Cycloconverter — Variable Speed AC Drive

Control is by variable frequency, and it is important to get the fundamentals right. An induction motor fed at frequency f runs at a speed of $n = \frac{60}{p}(f - s)$ rev/min, where s is the slip frequency and P is the number of pole pairs. By inspection of a family of induction motor torque slip curves for a range of frequencies it can be shown that provided the motor is worked in the linear range, where slip is small, then torque is proportional to slip.

The induction motor when fed with variable frequency thus works in a constant torque, variable horsepower mode, just like a shunt wound dc motor. The limit of torque is set by the torque slip curve of the induction motor.

This arrangement therefore provides the facility to start the fully loaded conveyor and run up to any value between 10 per cent and 100 per cent of full speed under controlled torque conditions. It thereafter permits continuous running at the selected speed against full load torque, with the facility to adjust the speed to any value within the permitted range.

DC Winder Type Motor
This type of motor is supplied and controlled by thyristor convertors, and has the main advantage that it is slow speed, and there is not therefore a need to specify a reduction gear unit.

Extremely long soft starts can be achieved, of several minutes duration, keeping starting torque at a very low level.

The motors can be run continuously at full load torque, from zero to maximum speed, and can therefore provide precise control under all conditions of loading. In addition, regenerative braking can be provided, with ramped slow down to precise limits, again under all conditions of loading.

These are just some of the various types of motor and control gear available, and each must be carefully assessed to determine the type most suited to the particular application.

It should be stressed that extremely rigid mountings must be provided for these high speed drives, to provide stability and alignment. Pulleys should be adequate in diameter, and of a proven design, preferably with turbine blade type end plates to provide resilience.

On uphill conveyors, where runback could occur, anti-runback devices should be specified. This is of vital importance, as a runback would cause serious damage, and could put the conveyor out of action for a very long time.

In view of this, it is desirable to fit an anti-runback unit directly to the head pulley, so that there is no weak link beween it and the belt, and provide a back-up unit on the transmission side.

For downhill, regenerative conveyors, a braking system must be employed, and stopping of the conveyor must be controlled through the brake system, so that there is no possibility of an overspeed condition starting to occur with removal of the generator load.

By careful choice of motor and control gear, and overall design of the drive unit, it is perfectly practicable to start up a fully laden conveyor with a starting torque that can be held to no more than 110 per cent to 120 per cent of full load running torque. Not only does this mean that the rating of the belt can be kept low, and significant gains in belt weight and cost be achieved, but the ratings of pulleys, shafts and bearings are all improved.

The Tensioning Gear
In any conveyor the correct tensioning of the belt is vital, as positive drive must be assured, and this is of particular importance in high speed applications.

There are many cases where it is perfectly practicable to apply the tension at the tail end of the conveyor, provided the conveyor is an uphill application, and the gradient is such that the slope weight of the belt is fully effective in inducing tension at the drive.

However, it is extremely questionable whether a high speed conveyor should be tensioned other than behind the drive unit, unless it is a downhill application where special consideration must be given.

Under all conditions of starting, running and stopping, correct tension must be applied to the drive. If this is not achieved, slip will take place and considerable belt damage can ensue.

Normally, the tension to be applied to the belt is determined by the maximum power requirement of the conveyor, and is derived from the well known formula $\frac{T1}{T2} = e^{\mu\theta}$.

On a conveyor which is designed to run at a single speed, this tensioning of the belt is only correct when the belt is fully loaded. At all other times, when intermediate loading is present the belt is overtensioned, since for any given drive the ratio of T1 to T2 is a constant.

I have already stressed the need to conserve the life of the belt as much as possible, and drawn attention to the fact that the effective life of a belt is largely determined by fatigue and wear.

It follows therefore that if for a large proportion of its life the belt is tensioned to a level which is only suitable for the 'peak load' condition, then a reduction in the life of the belt is being caused. This, to a point, is further aggravated by unnecessary wear, not only of the belt but all idlers and pulleys, because the conveyor is running at a speed higher than is required for the given load.

It is perfectly possible, however, to design a conveyor in which the slack side (T2) tension, can be automatically varied as the conveyed load changes, taking advantage of the fact that T1 and T2 are directly proportional. This is a relatively complex feature to achieve, and requires very accurate T1 and T2 tension sensors, whose outputs are compared, and if a variation + or − takes place, creates a positive or negative output whose value, and rate of change of value, is directly proportional to the tension change and rate of tension change.

This signal must be taken and amplified, and is then used to switch on an automatic tensioning device to either pay in or pay out belt to adjust the belt tensions to the change in load. The system must also be fully variable, and must respond at the same rate of change as the rate of change in load.

With such a system we have improved one of the factors that determine the life of the belt, but we are still left with a conveyor running at an unnecessarily high speed, still creating a maximum wear situation, and using more power than is required to run the conveyor economically. We therefore come back to the desirability of using a variable speed drive, and optimising on output.

With a variable speed drive, we need no longer worry about the complexity of varying the belt tensions to suit the load, but can specify a simpler and more straightforward tensioning arrangement, to apply *constant* tension.

As far as is practicable therefore, we adjust the belt speed to suit the load, keeping the tensions at that level required for maximum load and speed conditions. In doing this we reduce wear on belt and idlers, we conserve power, and we minimise belt sag between idlers, thus reducing flexure of the belt, movement of the material on the belt, and loading on the idler rollers. Remember, that the load on an idler set is not just the weight of belt and material. There is a tension component load imposed on the idler set due to belt sag, and the greater the sag the higher the loading will be. In addition to this, any lumps being conveyed can cause considerable impact to an idler set if the belt and load stream are subject to much flexure between sets.

I have stressed the need to apply the correct belt tension at all times, and this should be done automatically. There are a number of ways of achieving this, but all of them must be capable of operating at a speed which is commensurate with the rate of change of belt stretch (or contraction) which happens as the loading changes. Normally the biggest rate of change will occur during the starting up of a fully laden conveyor.

Even if the acceleration period is long, giving a low soft start, and keeping the overload value at a minimum; if the belt is long, then surging may occur, and this surging must be handled by the tensioning gear. The simplest and most effective method of automatic tensioning is the gravity weight type. Not only is it uncomplicated, it can be adjusted easily to give precisely the tension loading required, but is capable of operating at any rate to match the rate of change in belt tension, or slack belt being formed.

There are, however, many installations where it may not be practicable to fit a gravity loop, and in these cases power operated auto loops must be specified. These can be powered and monitored either electrically or hydraulically, but must be capable of operating at a speed which is commensurate with the maximum rate at which belt slack can be formed.

Where, for various reasons, it is not possible, or it is impracticable to make the auto loop function at this peak value (which on most conveyors will only occur during a loaded start), then a small travel gravity loop can be added into the system, or an accumulator arrangement incorporated to handle the surge situation, leaving the main function of the auto winch to deal with belt tension changes during normal running.

One major advantage of the powered auto loop, is that it can be made to lock-in the tensions in the belt when the conveyor is stopped. The induced tensions will re-distribute (slope weight tensions will not alter), reducing T1 and raising T2, and the auto tension unit must be isolated during initial re-start until T2 (locked-in) has reduced to T2 normal. The advantage of locking-in the tensions is that belt sag is minimised and this considerably reduces the torque required to restart the conveyor.

The T2 'locked-in' tensions can be considerably higher than the normal T2 value, and therefore all loop tension components must be designed to handle these 'locked-in' loads, particularly the braking system. However, there is a considerable advantage to be gained by designing the braking system to yield if the T2 belt tensions exceed the normal 'locked-in' value, which could occur if a malfunction developed in the system, and this yield feature would then protect both belt and conveyor.

At this point I must return to the gravity type take-up, and point out that this has one disadvantage over the auto loop, it does not lock-in the belt tensions when the conveyor is stopped, but allows the tensions to relax down to a mean value of T2, thus creating an undue sag situation on the carrying belt.

This can, however, be overcome by the introduction of a braked capstan unit, in the weight rope system, and a tension dynamometer. The capstan unit would normally free-wheel during running of the conveyor, but would be automatically braked on stopping the conveyor, thus holding the loop carriage and weights in position. Release of the brakes must not occur until locked-in T2 tension has reduced to T2, on re-start, this being monitored by the dynamometer.

In the automatic loop arrangement, the belt tension must be continuously monitored in order to adjust the tension unit correctly. Load cells can be employed, but these cannot easily be tuned to match the precise tension pertaining, and suffer from a considerable positive/negative change.

A spring system is far more sensitive, and can be readily tuned, and it is perfectly feasible to maintain tension to within ±5 per cent or less.

A further alternative is the constant torque tensioning unit that is powered the whole of the time the conveyor is running. This handles the slightest change in belt tension instantly, and is fully variable, and embodies many of the advantages of both the gravity and the normal auto tension loop. In special applications this type of tension unit can be combined with a gravity to provide instant adjustment under normal running, with the advantage of fast reaction on start-up with the gravity, and permitting a winch facility for belt storage and belt installation requirements.

I have written at some length on the tensioning arrangement, as this is almost certainly the most important part of the conveyor system, and a full appreciation of the requirements for this must be understood if a successful installation is to be achieved.

The Belt

The choice of belt to be used must be considered very carefully, being in most cases the most expensive part of the conveyor.

There are basically three types of belt, the solid woven or single ply type, the ply belt and the steel cord belt. Where high horsepower is allied to high belt speed then the choice will almost invariably be the steel cord type.

In any conveyor, and more particularly in a high speed one, weight is of paramount importance. The more weight, the greater the power required to run the conveyor. This therefore leads to the requirement to establish the highest practicable belt speed with the minimum belt width, taking all factors of materials, lump size, dust, etc. into account.

The safety factor of the belt is also of importance, not just from the point of view of ensuring that premature failure of the belt does not occur, but also from the point of view of keeping the belt weight low.

A further factor that must be considered when making the belt selection, and that is the stretch of the belt, and the characteristic of this stretch. The greater the stretch factor, the greater the problems of loop take-up and belt tensioning.

A solid woven belt, due to its crimped construction has a very high permanent set stretch characteristic that can be in the order of 3 per cent of its RMBT, and an elastic stretch of 2 per cent or more. In addition to this, the type of vulcanised joint that can be made rarely allows safety factors of less than 8 to 1 to be employed.

As stressed throughout this paper, the need to design for low soft starts, provide adequate belt support, and give good alignments, permits the safety factors of the belt to be kept at an economic level. For most standard conveyors solid woven belt safety factors of about 10 to 1 are more the norm.

The ply type belt is made from a variety of synthetic materials, and these determine its stretch characteristics and strength ratings. Generally speaking the elastic stretch is likely to be in the order of 1.5 to 2.0 per cent of its RMBT, with a small permanent set taking place within the first six months or so of running.

With this type of belt, the vulcanised joint is made by interlaying the individual plies, and this type of joint may well be stronger than that for a solid woven belt, and permit safety factors of about 7 to 1 to be considered.

With the steel cord belt, very high strength is achieved, and a belt can be tailor-made to suit the installation, by careful choice of the diameter, number, and material of the steel cords. Stretch of the belt is in the order of 0.2 to 0.3 per cent at RMBT, and there is virtually no permanent stretch. Vulcanised joints, with this type of belt, are immensely strong, and in static tests have strengths equal to that of the belt itself. However, nowadays it is recognised that fatigue strength is of greater importance, and considerable work is being done to establish ratings of fatigue strength that may lead to new and improved designs of joints.

Basically in a steel cord belt the cords are interlaced at the joint, but do not come into contact one with another. The strength of the joint is a mechanical/chemical interface between the steel cords, into which the rubber of the belt is impregnated, and the rubber matrix of the belt. There are several different types of joints, becoming more complex with increasing size of cord. The length of joint can also be varied, and may be several times the width of the belt.

With proper design of the conveyor drive and structure, safety factors for steel cord belts of 5 to 1 are perfectly practicable, and with further development may well be reduced below this level.

The life expectancy of the belts must also be considered. The cost of solid woven or ply belts is much less than that of a steel cord belt. However, their life expectancy is considerably less. Solid woven and ply belts, depending on the application, have a life expectancy of up to 8 years, whereas steel cord belts can be expected to last at least 15 years, and in many cases will continue to operate for up to 20 years or more.

The covers of the belts are also of concern, and must be chosen to provide good life, striking a balance on thickness, since very thick covers will increase the belt weight unnecessarily and the covers must also have good abrasion resistance.

In conclusion, in the writer's opinion, there are few high speed conveyor applications where the final choice of belt will be other than steel cord, unless the overall life of the project is limited to only 5 or 6 years. It must be remembered that while a steel cord belt may initially be expensive, when compared with the other types, if the projected life of the scheme involves the replacement of a single or multiple ply belt, then this is a very expensive operation, and with the initial cost of the first belt, may well be far more costly than having specified steel cord in the first place.

The Conveyor Structure and Idlers

There are several important aspects which must be considered here, in selecting the most suitable design of structure and idlers, for high speed applications.

Dealing first with the structure, this must be designed to permit very accurate alignment of the idlers to be made when the conveyor is installed. In this context the nature of the floor onto which it is to be mounted must be taken into account. If it is on stable ground, then suitable concrete mounting plinths can be specified for each stool. On the other hand, in a tunnel where an in-fill is to be used, the employment of cross girders, tied into the tunnel rings will provide a sound foundation.

In all cases, allowance should be made for packing the stools to permit initial alignment, leaving the idlers to be separately aligned.

Before leaving the question of the ground mountings of the structure, the ground may well have to be graded if there are undulations in the lines of the conveyor. Great care must be taken to calculate all convex curves. The concave curves must be sufficiently large to ensure that under all operating conditions, the belt is not lifted off the idler sets.

For convex curves the belt tension component acting on the idler sets must be taken into account in the idler roller load calculations. If large radii are not possible then it may be necessary to close up the pitch of the idlers to reduce the imposed loads on the idler sets within the radius.

A decision must be taken at an early stage whether a conventional type structure is to be employed, with the idler set mounted on the stringers. A preferred system is to design idler stands with the troughing sets mounted on each stand, and the returns on either every stand or alternate stands. In this type of conveyor structure, stringers are still required to act as guard rails along the conveyor and to carry protection equipment.

I have, in the course of this paper, stressed the need to keep belt sag as low as possible. This is particularly important for high speed conveyors. In this context therefore there is considerable merit in designing the conveyor to have the idlers very close pitched at the tail end, gradually increasing the pitching until a practicable maximum pitch is reached, the remainder of the conveyor having idlers pitched at this upper limit, except in areas of convex curves.

The mountings for the troughing and return idler sets should be so designed that these top and bottom sets can be individually aligned. It is of particular importance to reduce belt scrubbing and belt wander on high speed applications. To this end it may be necessary to machine the mounting faces for the top and bottom idler sets, provide shims for precise alignment of each set, as well as slotting, to provide adjustment in both the vertical and horizontal planes.

Generally speaking, some means of optical alignment is required, and this is usually carried out using master templates with a sighting device at each idler set.

The design of the structure must also take account of the requirements for mounting protection equipment, pull wire and communication systems.

The structure must also be designed so that there is no possibility of dirt and dust building up that interferes with the free running of the idlers.

Turning now to the idlers, the aim here must be to ensure the maximum life rating allied with the minimum resistance to turning, and the virtual elimination of maintenance requirements. To this end the idlers should be of the 'sealed for life' type, and should desirably have labyrinth type seals, where there are virtually no rubbing elements. Rubbing type seals create resistance, and naturally are subject to wear, gradually reducing the sealing efficiency.

Generally speaking, the most important element of the seal assembly of an idler is the grease. Not only does it lubricate the bearings, it provides an excellent sealing means. It therefore requires to be selected very carefully, and where extremes of temperature are likely to be experienced, should be of the type that has the least viscosity change at these extremes.

Extreme care in the choice of idler must be taken, as the idlers will largely determine whether the conveyor is or is not economic. Any unnecessary resistance to turning will add enormously to the power requirements and hence the operating costs of the conveyor. In addition the seals must provide for long idler life and the type of bearing chosen must be the best to ensure adequate load carrying and long life rating. It may well be more economical to choose an expensive, high capacity bearing, than a standard low capacity unit. If by choosing the higher rated bearing the number of idler sets, and mounting stools, can be reduced, the gain in power reduction requirements and the lower number of sets needed, may well offset the increased cost of the bearings.

For high speed applications, the accuracy and concentricity of the idler rollers is most important. Misalignment at high speed will cause the belt to be oscillated vertically, and this will cause vibration in the conveyor structure, and could lead to premature failure of the idlers, as well as increasing the operating power requirements. Therefore the concentricity of the idler rollers cannot be too highly stressed.

Finally, the idler sets must be designed to ensure the optimum alignment of the belt, and the minimum resistance to running. Return sets should preferably be of the V-type, and carrying sets of the 3 or 5 roll type. In both cases forward tilt of the wing idlers should be kept as low as possible, or even eliminated. Also, the weight of the moving parts in the idler rollers must also be kept low, consistent that is with long life expectancy.

The Feed Points

A conveyor may be fed at the tail end, or at both the tail end and at intermediate points. The design of the feed point will very largely determine the life expectancy of the belt.

If a belt is fed directly from a chute, then impact is involved, the material must be centralised, and the material has to be accelerated up to the full speed of the belt. With careful design of an in-line chute, good impact sets, and well designed skirt plates, a reasonable receiving unit/feed condition can be achieved.

However, with feeds coming on, other than in line, problems can be created if the material is not fully turned in line with the run of the conveyor, before leaving the chute, and in high speed applications the problems will be magnified.

With the best designs of feed and receiving units, it is inevitable that the belt will be subject to wear, due to impact of the material and more particularly due to the need for the belt to accelerate the material. Skirt plate rubbers cause belt edge wear and add considerably to power requirements, and all of these factors take on greater significance the higher the speed of the belt.

Therefore, for high speed applications, the feed should not be directly onto the main belt but onto an accelerator or intermediate feed conveyor. This unit would be mounted above and in line with the main conveyor, and run at a speed equal to or about 75 per cent of that of the main conveyor. For a variable speed conveyor the accelerator conveyor would also have to be variable, and would have a speed control loop interconnected with the main belt.

The advantages of the accelerator conveyor are that the material is already centralised before being transferred, therefore spillage and off tracking of the main belt is eliminated. The material is already moving at speed so that there is a minimum requirement to accelerate it, and by careful adjustment of height, impact can be greatly reduced.

The skirt plates on the main belt are clear of the belt and have no rubbers, therefore there is no rubbing and wear is eliminated, as well as a reduction in power requirements.

Finally, if there is a chute blockage, or a piece of steel or wood becomes jammed into the belt at a feed point, then the main belt is stopped, and in the latter case much expensive damage will be done to the belt. With an accelerator, the problems are limited to this unit only, a short, low cost belt only, is damaged, and the main belt can continue to run, and receive material if there is more than one feed point.

Monitoring

Having established all the various design parameters of the conveyor, and ensuring that the type of equipment chosen will endow the conveyor with a high reliability coefficient, and long life, it must nevertheless be recognised that malfunctioning can occur.

Any malfunctioning will be costly in terms of lost production, and may also be costly in damage sustained by the conveyor equipment. If a malfunction can be identified before a breakdown occurs, and/or a warning or trip-out occur before damage is done, then a costly, time consuming, repair exercise may be avoided, or reduced to a minimum.

It is necessary to examine carefully all the functioning parts of the conveyor and determine what the effect would be if any of these parts failed or malfunctioned. This examination will lead to two main categories of consideration, that where a malfunction will cause damage unless the conveyor is stopped immediately, and that, where if a warning is given at an early stage, the conveyor can continue to be operated, but remedial action must be taken at a convenient time.

All of these areas must be monitored, and the control or warning signals from the different monitors taken back to a control centre, and displayed on a suitable control console.

There is also a third category where control monitoring is necessary, and this involves a pull wire system, and a communications system. The pull wire is for the protection of personnel as well as the conveyor. If a pull wire is tripped, then it is of vital importance to the controller to know why and where the trip out has taken place. The facility of a communications system allows the nature of the problem to be transmitted to the controller immediately, and the necessary rescue or repair action put in hand with the minimum of delay.

On long conveyors, and/or where the conveyor is completely manless, it may be desirable to be able to identify on a monitor the zone in which a trip has occurred, and thus allow the controller to despatch help to the correct location, again with the minimum of delay.

Finally, the location of the conveyor may dictate that monitoring devices be installed to provide warning that faults have occurred which do not necessarily have a direct effect on the operation of the conveyor. For instance, in a coal mine it is necessary to protect against temperature rises in equipment, which could start a fire, but do not in themselves constitute a failure of the equipment. Smoke detection equipment is also necessary.

Let us now examine some of the equipment which requires to be monitored:

Pulley Bearings
These should be 'condition' monitored and also temperature. The 'condition' monitoring will indicate the start of deterioration in a bearing, and this can then be continuously checked, allowing eventual prediction of the time of failure. This allows for a planned replacement to be put in hand at a suitable shut-down time.

Temperature monitoring should indicate a warning that undue temperature is present, and a 'trip-out' if this is ignored and the temperature reaches a peak, but safe, level.

Other Bearings
Anywhere else where bearing failure would cause a shut-down of the conveyor. Idler bearings do not necessarily come into this category, but some form of temperature or audio monitoring may become desirable on certain special applications.

Hydraulic Systems
Monitoring of oil temperature, oil flow, oil pressure and oil level. Warning and/or trip-out must be determined for each particular case.

Lubrication Systems
Monitoring of oil temperature, oil flow, oil pressure and level. Where these service main bearings or other equipment which would constitute a major shut-down if lubrication failed, stand-by units may be desirable with automatic cut-in if a fault develops in the primary lubrication unit.

Belt Slip
Monitoring of drive and belt speeds should initiate a shut-down if a pre-set slip condition occurs, as if allowed to continue severe belt damage could result.

Torn Belt and Belt Wander
Monitoring at suitable points along the run of the conveyor.

Overload
Automatic trip-out if overload occurs. This could be on the drive motor directly, or could be a cumulative monitoring of all feeds at the various feed points.

Chute Blockage
This would constitute a trip-out on main discharge, but on feed points could be a warning if the blockage occurs on an accelerator conveyor.

Belt Monitoring
Belt slit monitoring is of vital importance, particularly on a high speed applications, and should be located at all feed points. It is of particular relevance on steel cord belts.

These are the most important areas for protection monitoring, but there are many others which must be given careful consideration in establishing the overall protection system. The system should also be interlinked so that there is no possibility of starting the conveyor unless all monitoring is in the 'GO' condition.

THE OVERALL CONCEPT

In the preceding sections I have dealt with the major constituent parts of a conveyor, but these must not be considered in isolation.

We are concerned with a complete system, and it will only be possible to achieve the objective of an effective, economic and viable transporter if the overall concept is kept firmly to the fore, and all design and functioning considerations are completely interrelated.

It is not possible to say any one part of a system comes first, it evolves if the project is tackled in the correct manner. This applies to any conveyor, but is of particular relevance in high speed applications, as a mismatching in the system will lead to failure, which may have far reaching consequences.

The customer, or the site, may well dictate that certain special aspects or equipment have to be considered in the overall scheme, but these must be properly integrated and co-related with the remainder of the equipment. A conveyor is a complete machine, not just a lot of pieces of equipment put together, with the vague hope that it will be all right on the day. A series of conveyors as a system must be treated in just the same manner.

The very nature of the subject allows me only to touch upon the major aspects which have to be considered, without producing an unacceptably long and laborious Paper. However, I hope to be able to show how all these aspects are put together by talking about a particular application, with which I have been involved since its inception. ●

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SELBY Conveyor

INDUSTRIAL
DIVISION
CONTRACTS
UNIT

Anderson
Strathclyde
Limited

47 Broad Street,
Glasgow G40 2QW, Scotland.
Telephone 041 554 1800 Telex 778109

TRAINLOAD OF COAL EVERY 20 MINUTES

ANDERSON STRATHCLYDE are designing and developing one of two drift conveyors for the British National Coal Board's Selby Mine project. A declaration of intent to accept Anderson Strathclyde's proposals has been made by the NCB, although contract details have yet to be finalised.

NCB
Plan for Selby

SELBY mine, the British National Coal Board's largest and most ambitious project, is scheduled to yield more than 10 million tonnes of coal a year by 1986.

The coalfield, which lies beneath the Vale of York, covers an approximate area of 100 square miles between the city of York and the town of Selby. It has an estimated total extractable reserve of 300 million tonnes of coal.

Mining complex
– five shafts

Selby mine is really a mining complex of five shafts down which men and materials will be taken to the coal face. The total output of the five mines will be brought to the surface by conveyors in two main spine roadways which are 14.5km long and rise to the surface by a 1 in 4 drift.

The mine will eventually employ [illegible] miners, 3500 of them below ground, and most of its output will go to three neighbouring power stations via a merry-go-round train system.

Selby project is the result of an intensive search for new coal reserves launched in 1972.

The conveyors will bring all the coal to the surface from five new NCB pits at Selby in Yorkshire.

The Anderson Strathclyde conveyor, most powerful in the world, will be capable of delivering a trainload of coal every 20 minutes. It will be installed in the southerly of two underground main parallel spine roadways and will incorporate several revolutionary features.

Steel cord

Close on 15km long, the conveyor will bring coal to the surface from about one kilometre below ground level. A steel cord belt, 1300mm wide and weighing over 2500 tonnes, will travel at speeds up to 30km an hour. It will be capable of delivering 1800 tonnes of coal an hour from the farthest and deepest point of the conveyor or 3200 tonnes an hour from nearer the surface.

When maximum tonnage is not needed the speed of the conveyor can be controlled to produce only the required tonnage, so that wear and tear is minimised. This is a particularly attractive feature of the Anderson Strathclyde product as the belt represents a significant part of the total installation cost.

The two spine conveyor roadways at Selby run from the south-westerly corner of the coalfield to the north-east. Lateral underground roadways driven in the seam link the five mines to the tops of 11 strata bunkers. Coal is fed into these bunkers then subsequently extracted by bulk density feeders and fed on to the main spine roadway conveyor by accelerator conveyors.

The conveyor belt comes out of the spine roadway up a 1 in 4 drift, through a 4500 metre radius. It passes up the main ramp, along the horizontal section to the main drive arrangement where the coal is discharged.

World's most powerful conveyor

On its return journey the belt passes through a cleaning system and around a bend pulley. It travels up to a moving loop carriage, across a permanent vulcanising station, over a pulley adjacent to the portal and on down the drift again

On the same drift there is a rope haulage system for transporting men and materials underground.

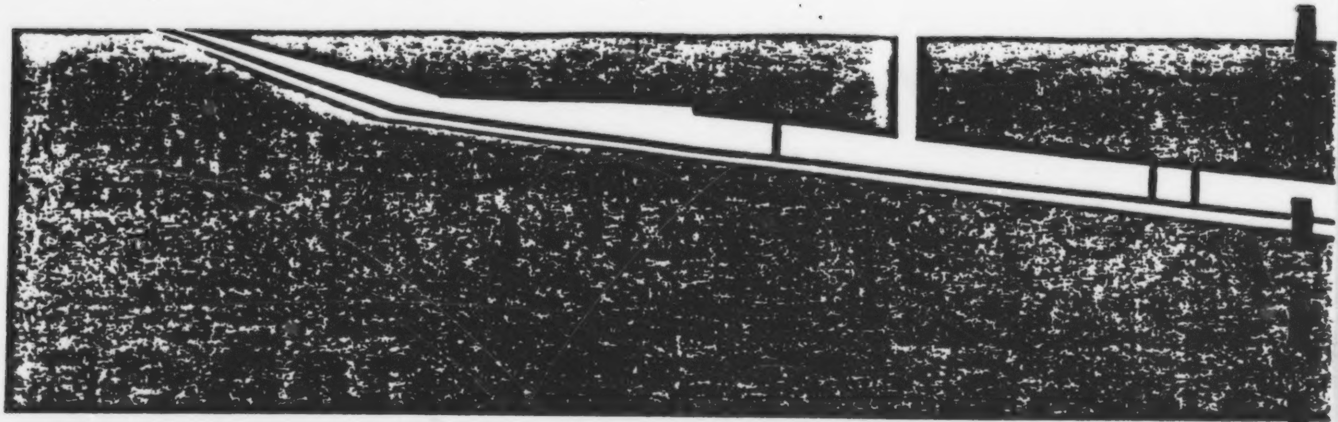
Novel features

One of the many novel features of the conveyor is the method of installing new lengths of belt Because it is due to be extended progressively from a length of about 3000 metres to 15,000 metres it was felt there was a need to have a permanent method of adding new lengths of belt and vulcanising them to the existing belt.

New reels of belt weigh around 40 tons and a crane has been introduced to lift the reels

Continued on Page 2

ANDERSON STRATHCLYDE-TOMORROW'S MINING WORLD



Trainload of coal every 20 minutes

(Continued from Page1)

and place them on to the framework which then locates them beneath the conveyor.

When an extension is required, the end of the reel is unwrapped and moved on to the vulcanising table. It is accurately aligned with the belt already on the conveyor and is vulcanised into place with the aid of the loop carriage and four clamps.

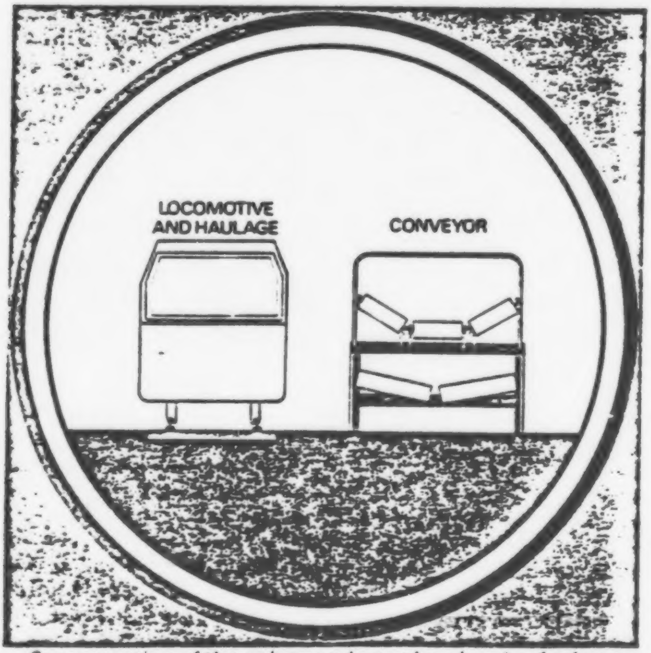
By means of the moving carriage, new belt is drawn from the reel and lowered into the drift about 50 metres at a time, until the other end of the reel reaches the vulcanising table.

While this operation is in progress the existing belt has been anchored at one of the other clamps and is now ready to be joined to the tail of the new piece.

Although the lower carriage is used to install the belt it also applies tension to accommodate belt stretch on start-up and during normal running when the load varies.

Tension

At start-up the 150 tonne gravity weight will be positioned at the mid point in its 22 metres deep pit, but as natural belt surges take place, the weight can keep the required tension in the belt. Once stabilisation has been achieved the final tensioning of the belt is done by means of a constant-torque capstan.



Cross-section of the spine roadway showing the Anderson Strathclyde conveyor and haulage system running alongside. The roadway diameter is 5.3m.

Under normal circumstances when the conveyor stops, the maximum running tension will be locked-in to prevent the belt sagging between the idlers and thus making starting difficult. Positioning of the weight is achieved by the weight adjustment winch which was used for the positioning of the carriage when the belt was originally installed.

The conveyor is driven from the head end by a single drum powered by two directly-coupled DC motors to the NCB rationalised winder motor design. This main drive is unusual in that the drive pulley is a single hollow forging over two-and-a-half metres diameter at the drive drum and over one-and-a-half metres at the bearings.

The motors — two of the largest DC models ever installed by the NCB — have a total power of 10,000kW, and are of the bearing-less type, with armatures directly coupled to the ends of the drive drum shaft.

Acceleration

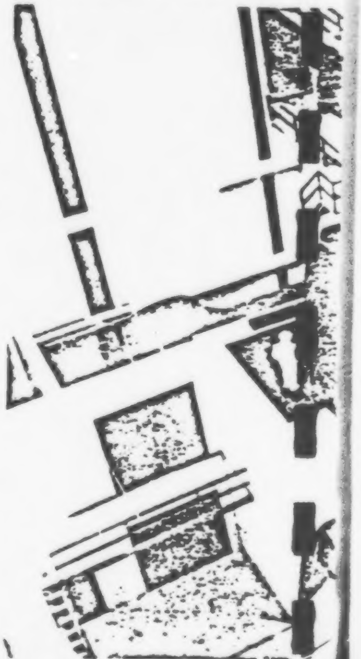
Operating continuously at any speed between 0 and 60rpm, the motors permit the conveyor to be started gradually with an acceleration time of at least four minutes. This limits the starting torque to 106 per cent of full load conditions.

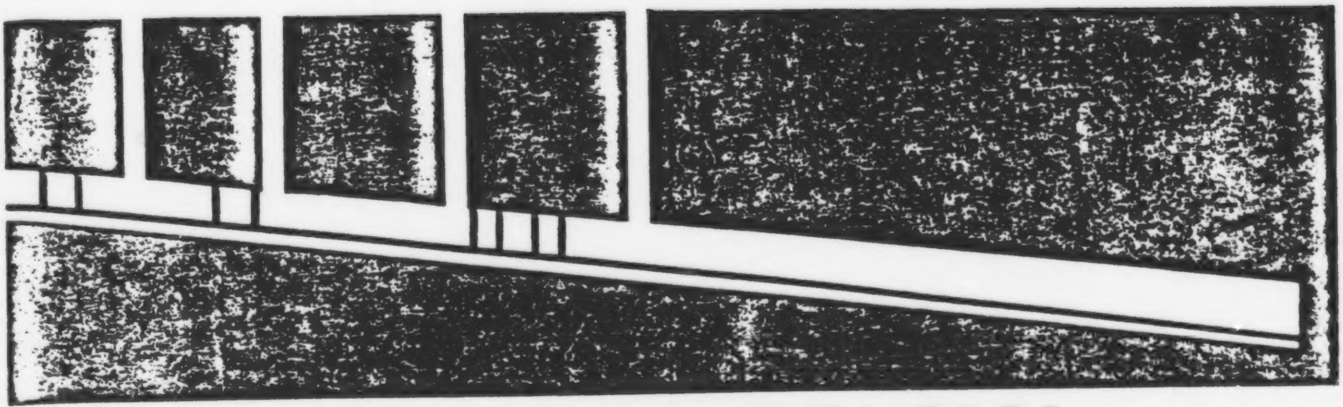
The motors allow the conveyor to be run at the minimum speed commensurate with the various loading conditions and they can be computer controlled in conjunction with the 11 feed points.

Air for the motors com from outwith the building. It enters beneath the moto through a large louvre and c then be circulated either to upper or lower fan through a pair of automatically advan ing roll-type filters.

The air is brought throug the foundations into the front of the motor, passes throu the motor armature and start and is fed below to outlet ducts. The ducts are also used to carry the cables from t main control room to t motors.

Anderson Strathclyde's successful design is the result close co-operation with t French company Realisation Equipments Industriels who have designed and install long distance surface conveyo in various parts of the world.





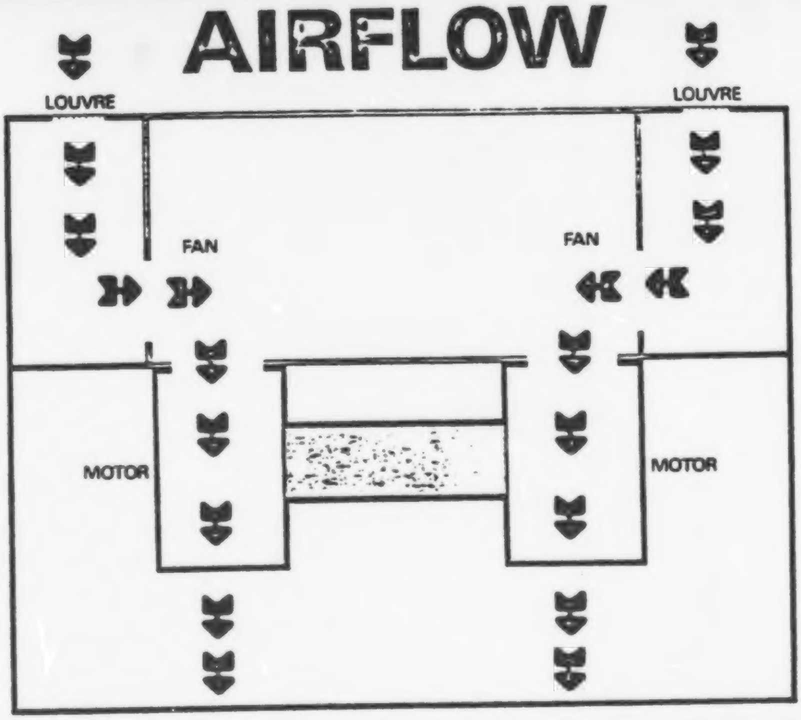
ABOVE – A cross section of the mine showing the five shafts and 11 loading points. The roadway is 14.5km long and is 1km below ground at its deepest point.

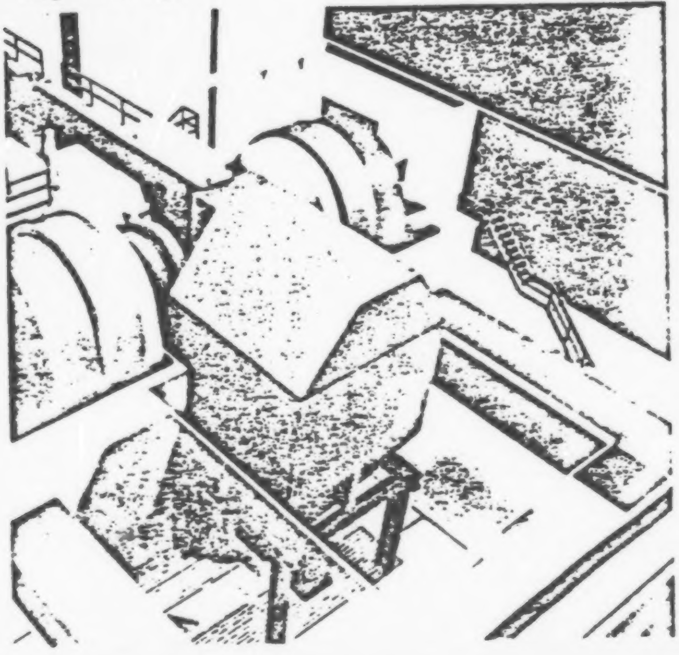
☆

RIGHT — A diagram shows the airflow to cool the two electric motors which drive the conveyor.

☆

BELOW — A model of the drive arrangements shows the two 5050kW electric motors coupled to both ends of the drive shaft and the discharge chute feeding on to two surface conveyors. In centre foreground, one of the cooling air ducts as designated by the arrows at the foot of the diagram on right.



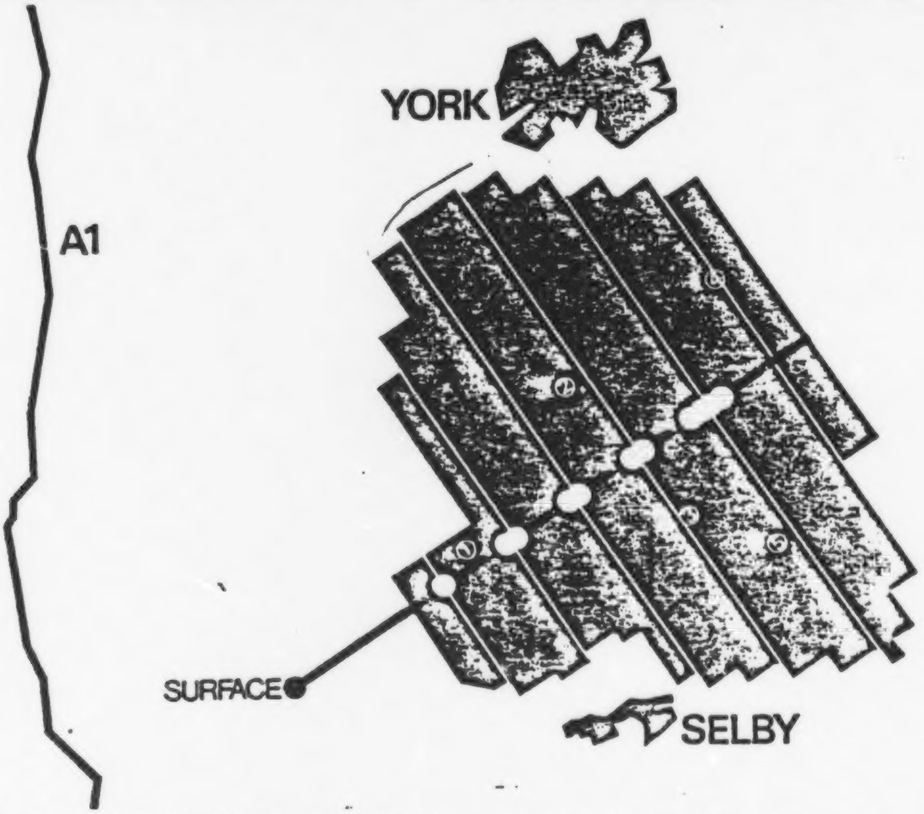


PLUS POINTS

POINTS in favour of handling Selby as a single-flight conveyor include:

- All drive and control equipment is in one place. There is no problem with sequence starting as there would be with a series of conveyors.
- The underground portion is completely contained within the tunnel and there is no necessity for high-voltage underground cabling to intermediate drives.
- With drive and control on the surface there is no need for flameproof equipment.
- The heat generated by underground drives — in what is already a fairly high temperature environment — is avoided.
- The progressive extension of the conveyors as the tunnels advance is made as simple as possible by moving the tail pulley unit and adding further idlers and belting. This means that downtime is kept to a minimum.
- All belting is installed and spliced on the surface, avoiding the problems associated with handling heavy belting underground.

10 MILLION TONNES A YEAR



The Selby mine showing the five shafts, the main spine roadway, 11 loading points and, running into the spine, the cross-roadways along which coal is conveyed to the loading points.

Facts at a glance

DRIVE

Motors	Two x 5050 kW DC Winder Type 0-60 rpm 'E' frame
Drive Drum:	2.674 m single with replaceable steel liners.
Tension Unit:	Combined gravity and powered automatic loop take-up.

BELT

Width:	1300 mm
Construction:	Fire resistant steel cord ST7100
Speed:	Infinitely variable up to 8.4m/sec.

STRUCTURE

Construction:	Independent stools
Spacing:	6M pitch closing over last 2Km in graded steps to 1.5M pitch
Trough Idlers:	3 roll 30° on each stool
Return Idlers:	2 roll 'V' type on each stool

FEED ARRANGEMENTS

Accelerator Conveyor:	Variable up to 6.3M/Sec.
Receiving Section:	3 roll close pitched impact idler sets.

TAIL END

Drum:	Single unlagged rail mounted
Tension Unit:	Manual winch

BELT CLEANING

Washing:	Recirculating water syste
Drying:	Vacuum and squeeze rollers

MONITORING	To suit planned compute ised control.

Huwood Booster Belt Conveyors

One of the major problems associated with the planning of a heavy duty conveyor installation, is in determining the maximum power necessary to cater for the ultimate mining requirements, and relating this to the most suitable drive equipment and conveyor belting. Situations do arise, where, following installation of the conveyor, it has been necessary to increase the length of the conveyor or due to improved extraction methods, the conveyor capacity has increased. In these circumstances, it has generally necessitated the introduction of :-

(i) A larger drive unit involving additional excavation work.

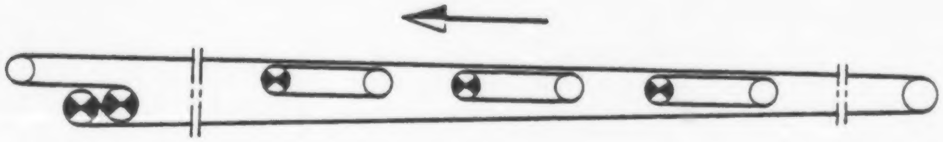
(ii) An entirely new conveyor belt, to suit the demands of increased tension and capacity.

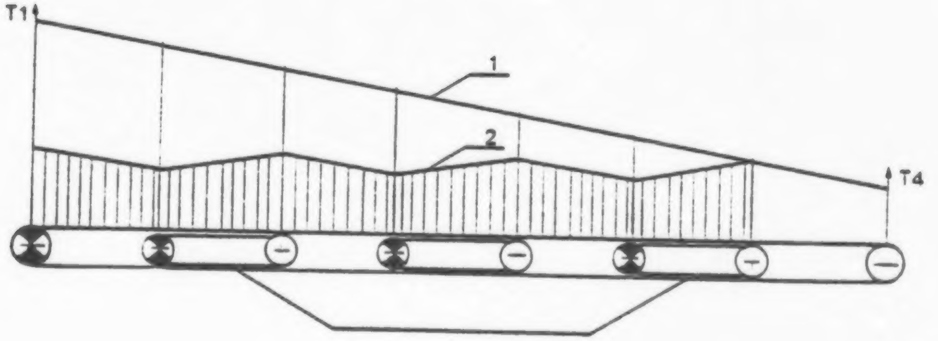
(iii) New delivery and return end equipment and L.T.U., to accommodate the higher tensile belt.

These changes obviously incur a considerable capital expenditure, apart from the inconvenience involved in the installation of new equipment.

A solution to this problem, in the form of a 'Booster Drive', has been developed by Huwood Limited.

The Booster Drive is designed as an intermediate drive, which can be installed in a predetermined position within the conveyor, power being transmitted by an endless driving belt which is in contact with the carrying belt. This frictional contact imparts the additional power requirement to the original belt.





BELT TENSION DISTRIBUTION WITH MULTIPLE BOOSTER DRIVES

The tension diagram illustrates the tension requirement of a high horse-power single flight conveyor system (curve 1), compared with that of a system incorporating several intermediate Booster Drives, (curve 2).

The first installation of a Booster Conveyor in the United Kingdom was introduced at the Royston Drift Mine, in the Barnsley Area of the National Coal Board. The mine was officially opened by the N.C.B's Chairman, Sir Derek Ezra, in September 1976. As a result of the subsequent productivity achievments, it was decided to implement plans to increase the overall output of the mine in 1978.

The surface drift conveyor, which was installed at the time of opening, was designed to the following specification:-

Conveyor Length – 2265 feet

Conveyor Lift – 375 feet

Conveyor Capacity – 500 tons per hour

Belt Width – 42 inches

Belt Type – N.C.B. Spec. Type 8 solid woven.

Belt Speed – 500 feet per minute

The conveyor selected for this installation was produced by Huwood Limited, and powered by a Type 'TB300/500' Drive Head, fitted with a 385 H.P. electric motor. A motorised take-up unit was situated at the rear of the drive, to accommodate the permanent stretch and a gravity type take-up at the bottom of the drift, for the elastic stretch.

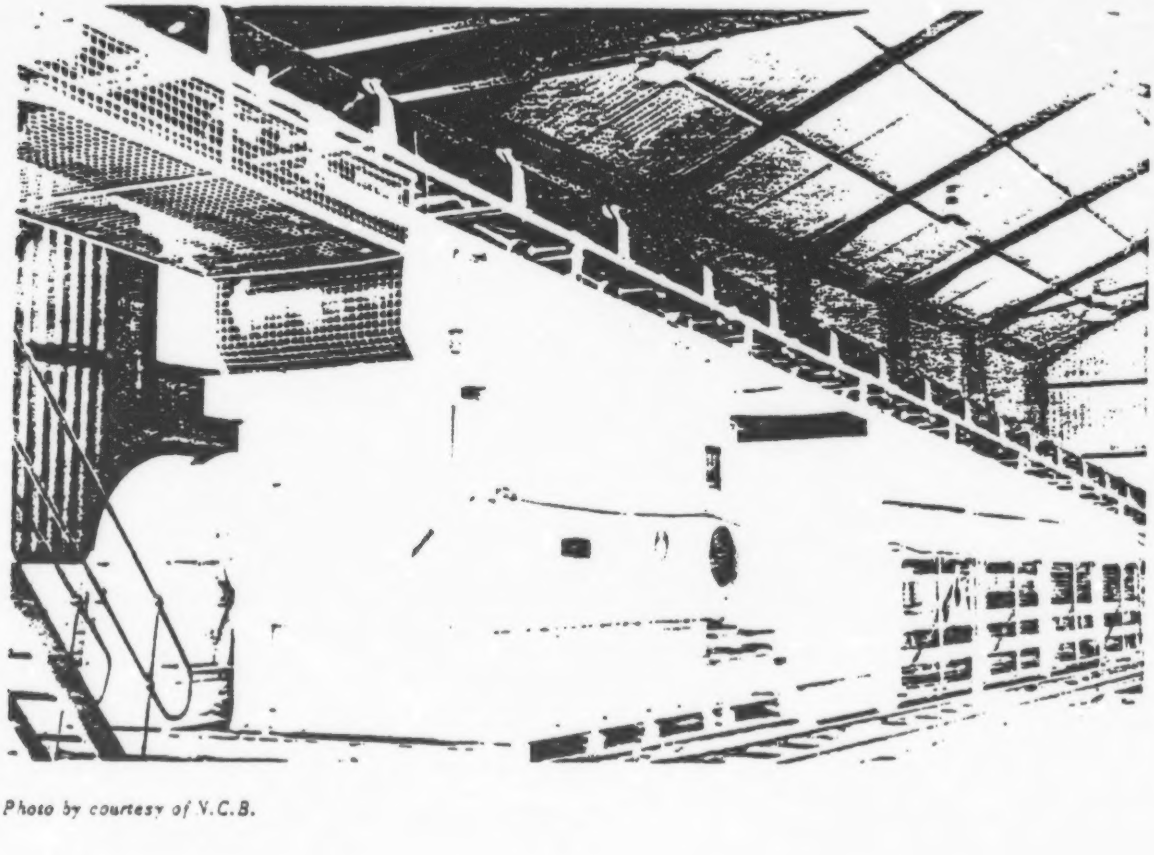


Photo by courtesy of N.C.B.

A decision was made to increase the duty of the conveyor to 1000 tons per hour, which necessitated increasing the 42" belt speed to 600 feet per minute and the drive requirement to 580 horse-power.

The following alternative options were then considered :-

1. Replacing the existing 385 H.P. drive with a new 580 H.P. drive, and the existing Type 8 belt with a heavier duty Type 10 belt. The delivery and loop units would also have to be replaced to suit the increased tension requirements.

2. Shorten the existing 385 H.P. conveyor, and install an additional 300 H.P. conveyor, which would involve the introduction of a tandem transfer point.

3. To retain the existing 385 H.P. drive and Type 8 belt, and install a Booster Drive in the appropriate position in the drift. The conveyor length would be unchanged and the need for a transfer point obviated.

The Booster System was selected and commenced operation in January, 1978, and has continued to operate with satisfaction. The following advantages emerge from our experience at Royston Drift Mine :-

1. Considerable cost saving due to :-
   (a) Retention of original belt.
   (b) Reduced investment on drive and ancillary equipment.
   (c) Minimised excavation for drive equipment.

2. Standardisation of equipment incorporated within main and Booster drives, which minimises the spares holding.

3. Elimination of transfer point/s, thus preventing degredation of material, hazards relating to airborne dust and manpower requirements often associated with transfer points.
   This could also result in an improved belt life.

4. In the event of belt man-riding, a continuous ride will be available on the top belt, which reduces the quantity of boarding and alighting stations and facilitates the electrical control system.

The success of this installation has led to the adoption of this unique concept of conveying by the National Coal Board and has resulted in further orders being placed for Booster Drives.

In conjunction with the conveying equipment, Huwood Electric, the Electrical Division of Huwood Limited, are specialists in the design and manufacture of control and monitoring equipment for use in mines, and as Booster Conveyors require specialised control and monitoring equipment to ensure smooth starting and load sharing between Drive Units, their expertise is particularly valuable.

Experience already gained in the British Isles, enables Huwood Electric to offer with complete confidence, full monitoring and control systems as part of the Huwood Limited pack of Booster Belt Conveyors.

The timing of sequence starting is essential for the operation of Booster Drives, and this feature is built into the Huwood control unit. Two or more units are required to control and monitor the Main and Booster Drives, these units are constructed to meet the following stringent requirements:

1. Due to the solid state electronics compact units are designed, reducing underground excavations.
2. Huwood Electric's underground experience ensures designs to be reliable in wet and dusty conditions.
3. The correct operation of the system is monitored not only at the delivery end of the conveyor but also wherever the Conveyor is Boosted.
4. The Huwood Electric Control System can be used in conjunction with existing operational systems.

BOOSTER BELT CONVEYOR AT ROYSTON DRIFT MINE

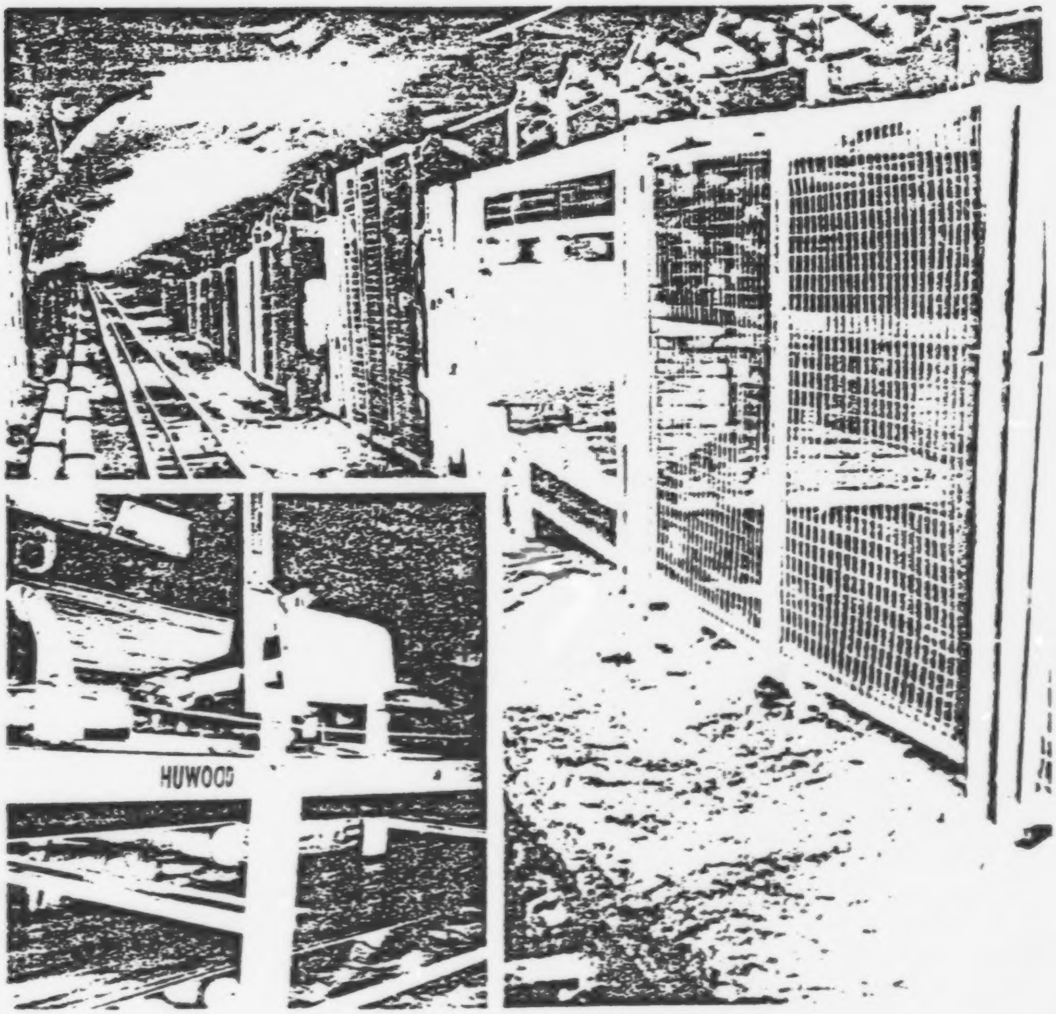
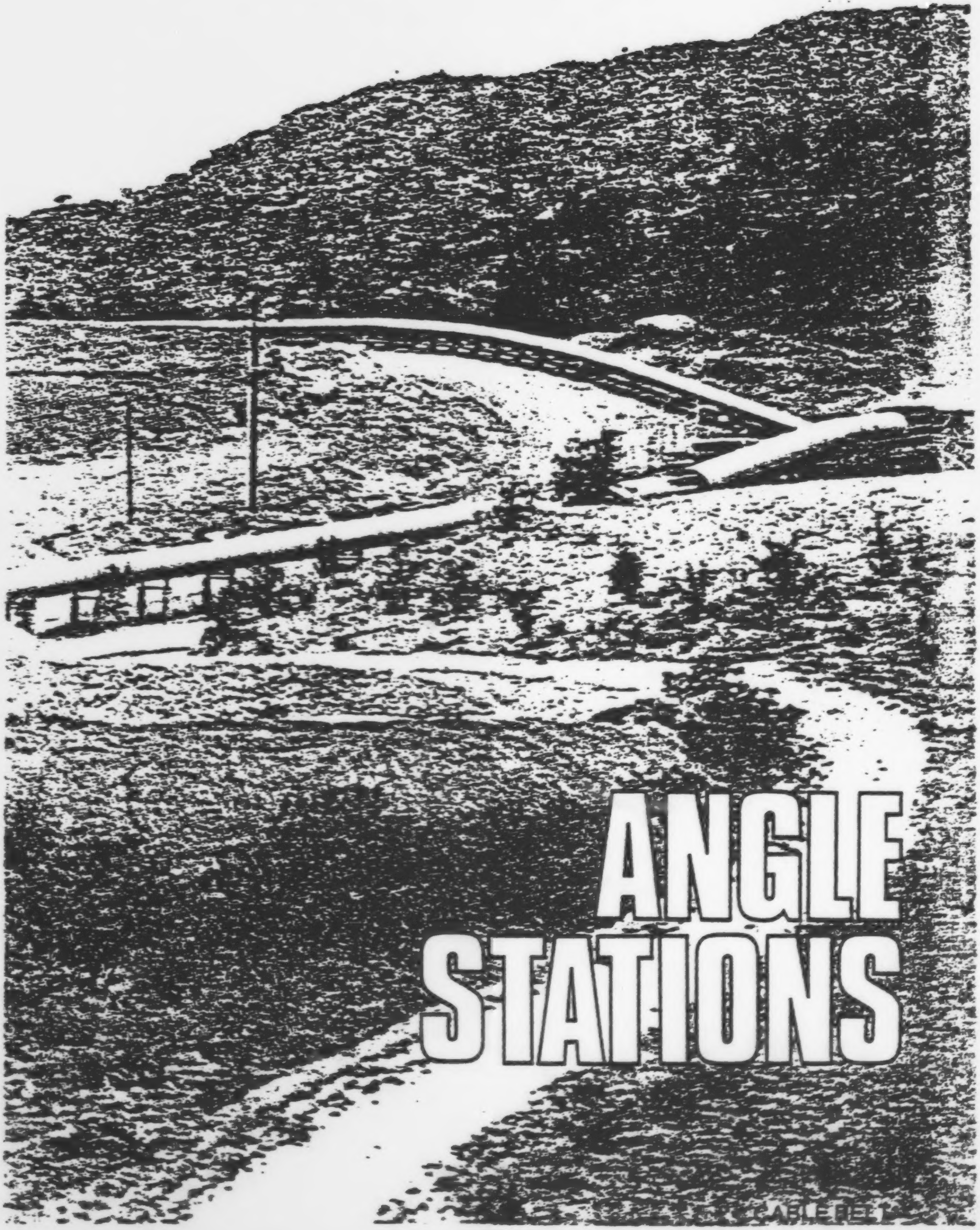
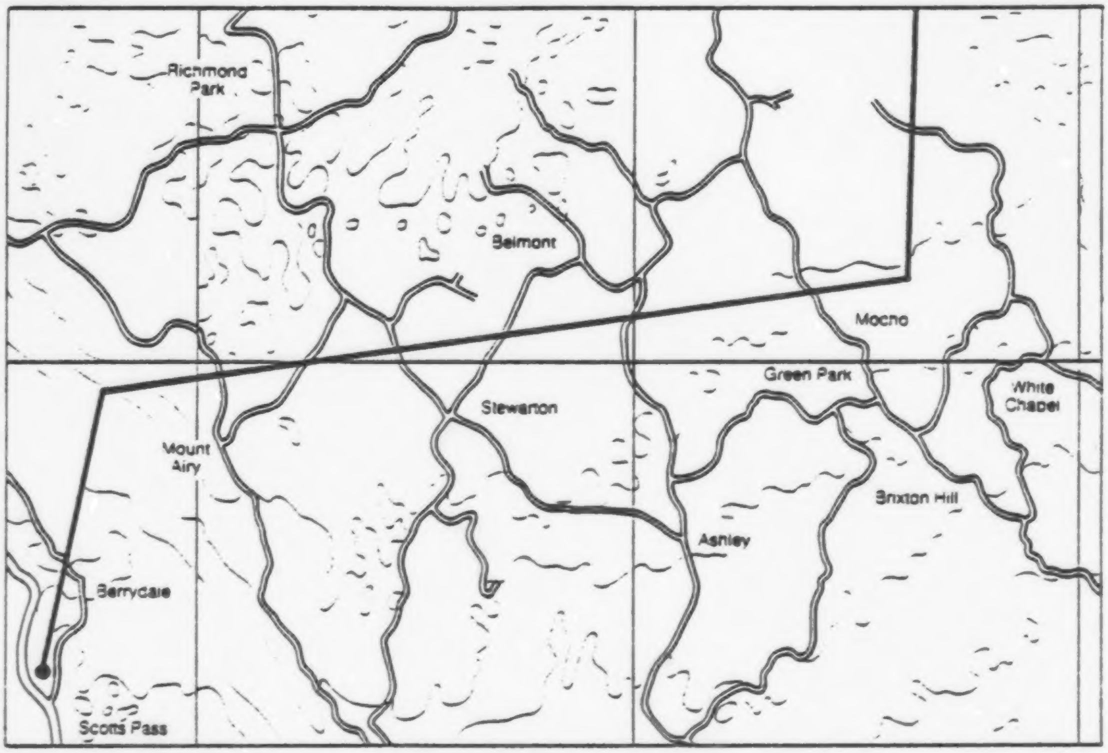


Photo by courtesy of N.C.B.

ANGLE STATIONS



The shortest distance between two points is NOT ALWAYS a straight line.


Richmond Park
Belmont
Mocho
Green Park
White Chapel
Stewarton
Mount Airy
Brixton Hill
Ashley
Berrydale
Scotts Pass

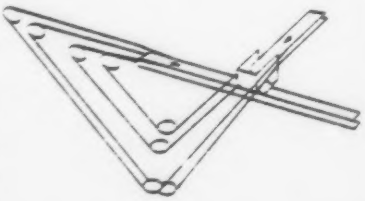
Long conveyors often cannot achieve the ideal of a straight line due to restrictions imposed by the terrain, obstacles en-route or the right-of-way that can be obtained.

With conventional troughed belt conveyors such a route is followed by using a series of shorter conveyors – one conveyor for each straight leg of the route. Each conveyor has to be individually driven and sequenced, necessitating power being led to each drive location along the route and compounding the possibility of the system malfunctioning.

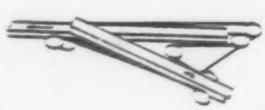
Where changes in direction are too sudden to allow curving the conveyor, the CABLE BELT® system can, by the use of *Angle Stations*, readily overcome such restrictions and still require only *one drive unit*.

To cover all conditions, three basic types of *Angle Station* have been evolved.

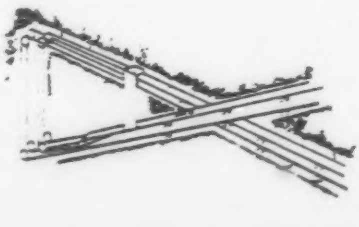
Type 1: Horizontal Interchange



Type 2: Horizontal Interchange



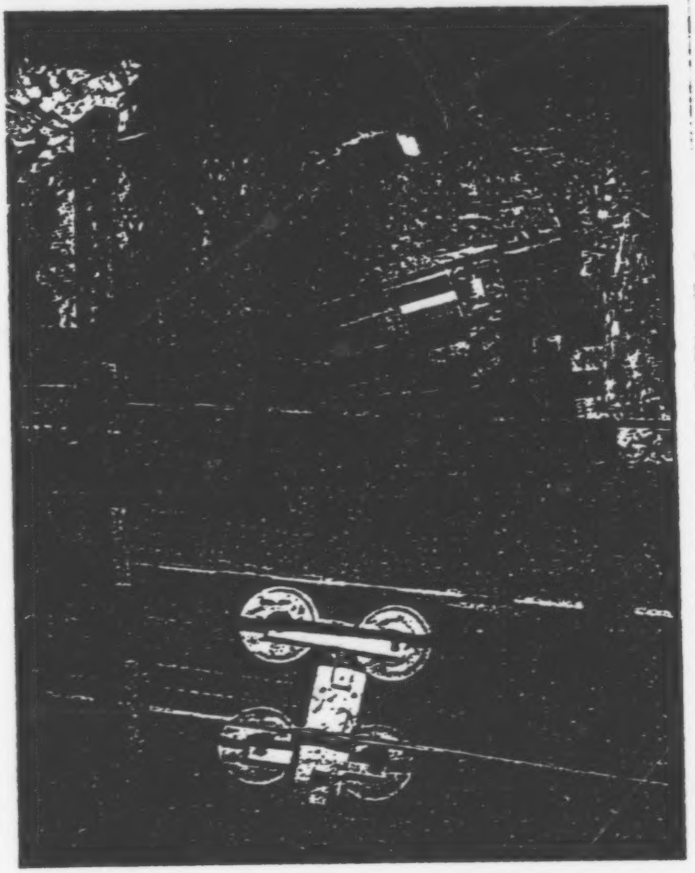
Type 3: Vertical Interchange



The Angle Station is composed of a Discharge Unit, a Loading Frame, Belt Tensioning Units and cable deflecting pulleys. The belt on the incoming conveyor leg discharges onto the belt on the outgoing conveyor but the Drive Cables pass through the Station to a Single Remote Drive Unit and are common to both belts.

The discharging belt, with material, is raised from the Driving Cables on idler rollers at the Discharge Unit and discharges over the drum into a chute to the Loading Frame which feeds the material onto the receiving belt. A series of pulleys deflect the Driving Cables to the new direction and align them with the cable shoes on the receiving belt.

Both belts are individually tensioned by gravity weight systems.



A Type 3 Underground Angle Station at S.A. Manganese, Wessels Mine, Hotazel, on a CABLE BELT® conveyor handling Manganese Ore

An aerial view of a 6 Km CABLE BELT® system carrying iron ore at the Scunthorpe works of the British Steel Corporation. The system includes three Angle Stations, two of which can be seen in the photograph.

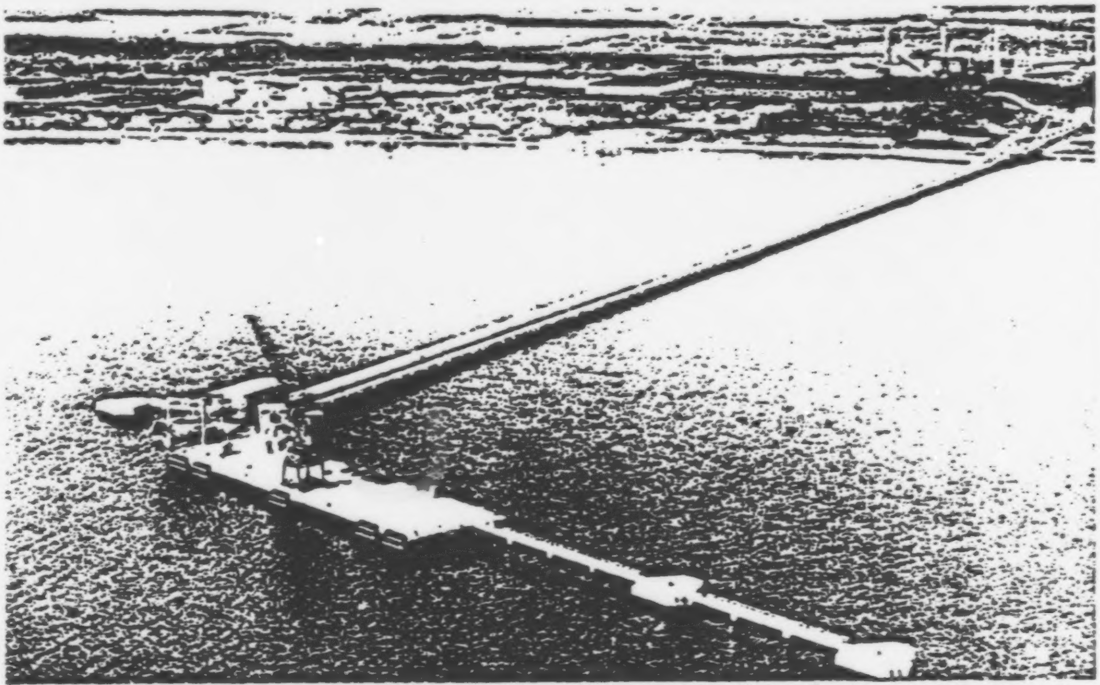
Any angle of bend up to almost 180 degrees can be accommodated as each Angle Station is purpose built. Particularly difficult routes can be accommodated by using a series of Stations.

An operator is not required at an ANGLE STATION

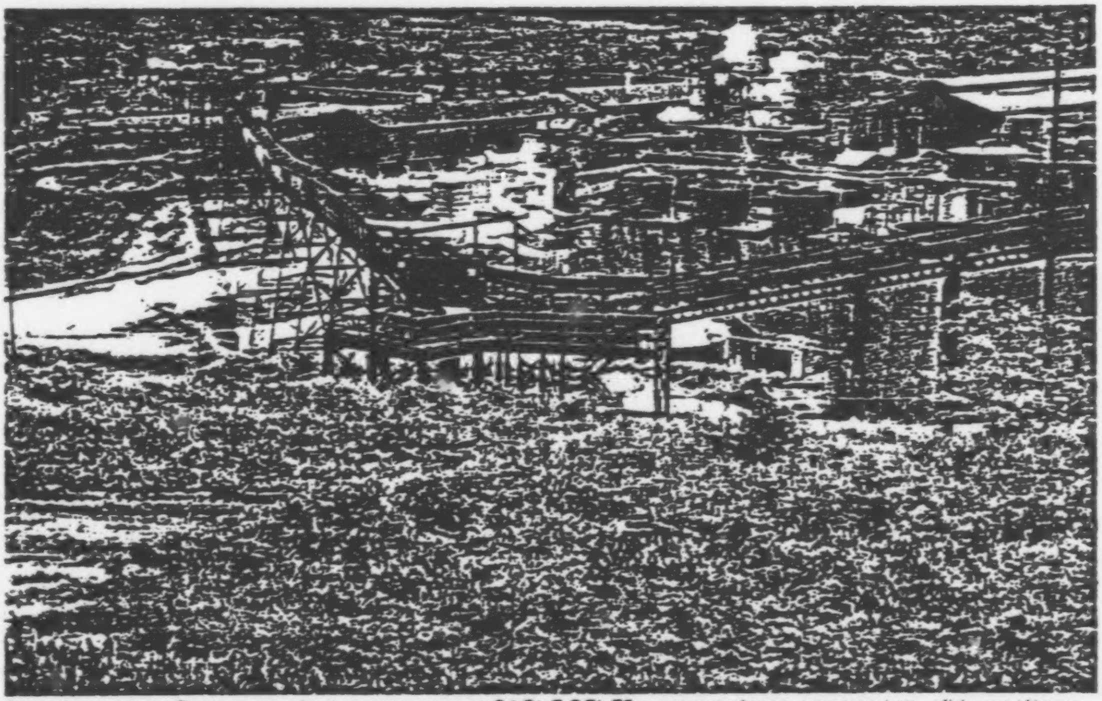
– the complete CABLE BELT® system is operated from a central control.

The unique Angle Stations on the CABLE BELT® Conveyor mean–

- ✲ No separate Drive for each leg.
- ✲ No sequence control problems.
- ✲ No additional power cabling.



A CABLE BELT® Conveyor, installed at British Aluminium Ltd., Invergordon Works, Scotland, incorporating an Angle Station, conveys alumina from ship to plant.



One of two Angle Stations installed on an overland CABLE BELT® system at Alcan Jamaica Ltd., Kirkvine Works, Jamaica, and handling Bauxite.

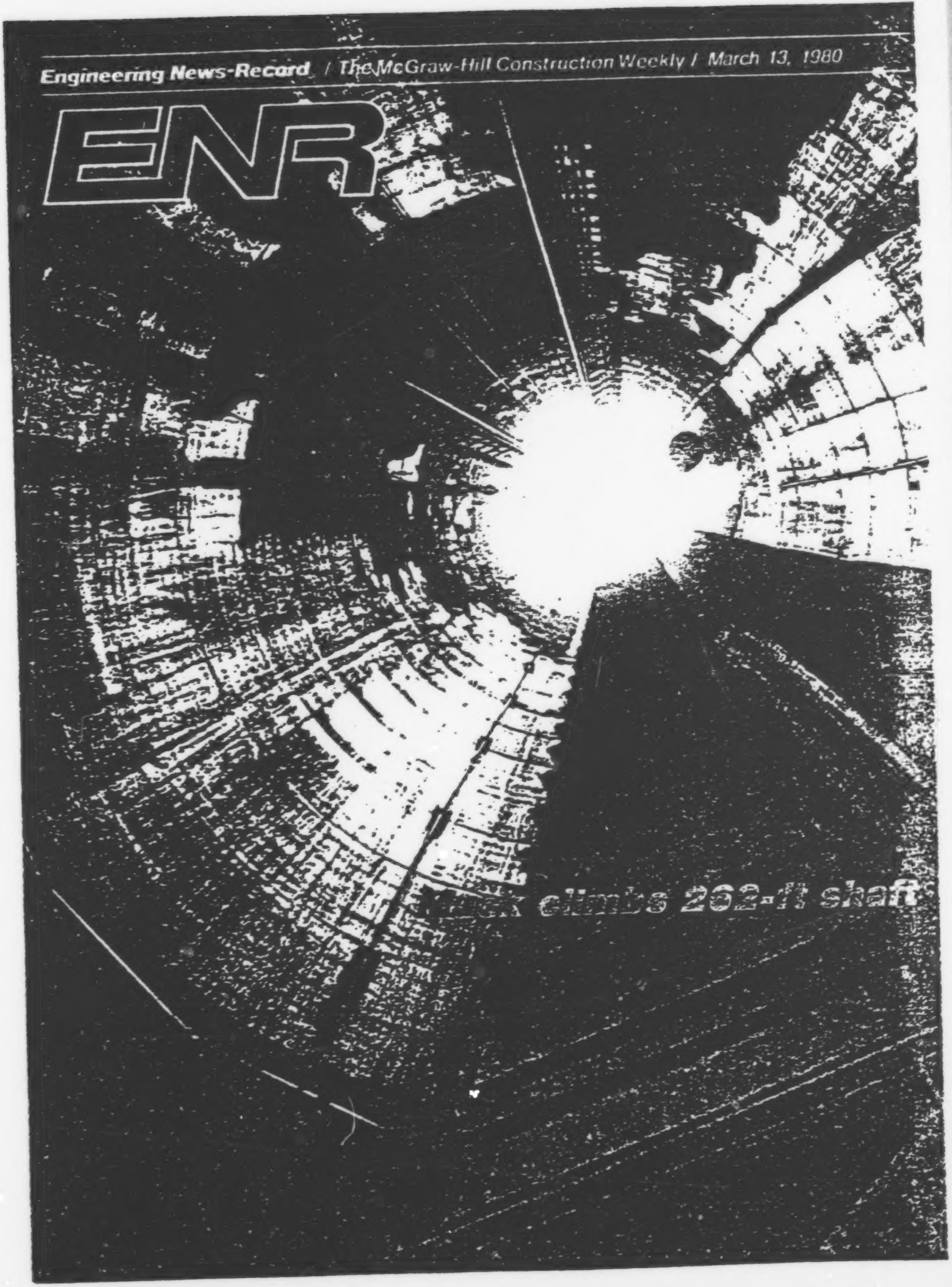
Angle Stations are also used on underground systems.

Frontispiece: An Angle Station on an overland CABLE BELT® Conveyor at Anamax Mining Company, Tucson, Arizona, U.S.A. handling Copper Ore.

CABLE BELT® is a registered trade mark of the Cable Belt Group of Companies.

Engineering News-Record / The McGraw-Hill Construction Weekly / March 13, 1980

ENR



[illegible] climbs 262-ft shaft

SCHOLTZ · E F S CORPORATION
P. O. Box 353
Clinton, Connecticut 06413
Phone: 203 / 669-0504

Machines muck 3,000 tons an hour from 110-mile sewer tunnel project

Vertical conveyor speeds mucking on $2.8-billion job

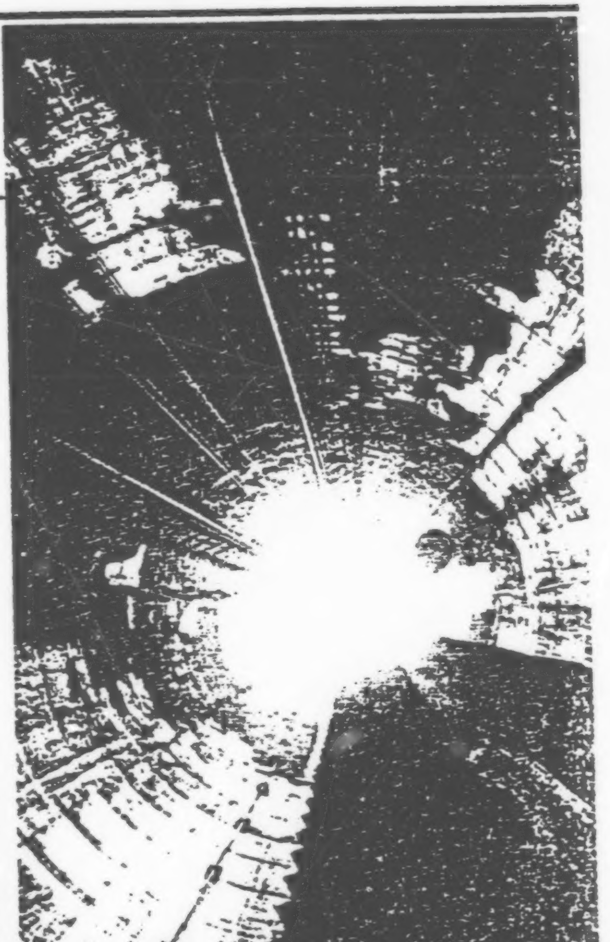
Trains haul muck to hopper where cars are flipped, dumping spoil onto first section of conveyor system.

A novel vertical conveyor belt is proving to be the fastest and most efficient method among the various mucking systems being used on Chicago's 110-mile, $2.8-billion deep tunnel and reservoir project (TARP).

With a capacity for handling up to 1,100 tons per hour of muck, nearly twice the quantity most conventional hoisting systems can handle, the vertical conveyor has been so successful that the joint venture using the system now touts it as the answer to the ever present tunneling problem of spoil removal. "With its remarkable performance record, I don't see why a contractor would choose a different system at this point. The vertical belt represents a quantum leap" in solving mucking problems, says E. O. Mixon, project manager of Kenny Construction Co., Wheeling, Ill., a member of the TARP joint venture using the system.

The vertical conveyor is being used on a $107-million, 4.8-mile portion of TARP by the joint venture of Kenny; Paschen Contractors, Inc., Chicago, the primary contractor; and Morrison-Knudsen Co., Inc., Boise, Idaho. It is currently handling about 700 tons per hour of muck. The conveyor, being used in the U.S. for the first time, was designed by Conrad Scholtz AG, Hamburg, West Germany, a manufacturer and supplier of

Inclined belt transfers muck into 569 pockets on vertical belt.



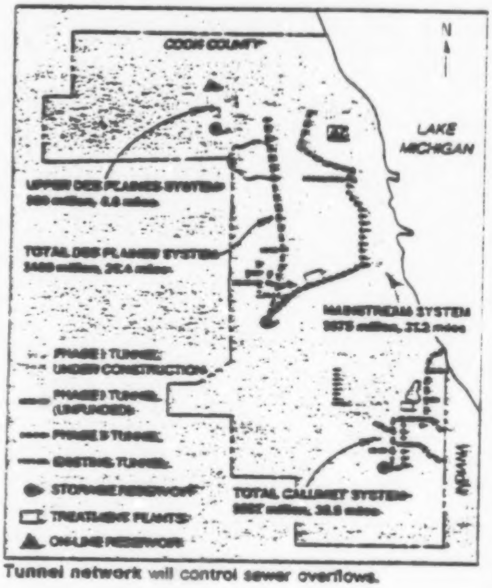
Enclosed belt moves 700 tons of muck per hour.

conveyor belts to the European coal mining industry, and Zokor Corp., Aurora, Ill., the manufacturer of the belt's trailing gear and support structure.

Fast climbing conveyor. The conveyor's belt is 631 ft long, 63 in. wide and carries the muck to the top of the 262-ft-deep construction shaft in 569 pockets at a speed of about 450 ft per minute. The 32-in.-dia pockets on the vertical belt are formed by 16-in.-high sidewalls and replaceable, preformed rubber scoop dividers.

The conveyor system is enclosed in a 7 x 6-ft metal casing in the shaft as a safety measure and to keep it warm during cold weather.

The mucking process, however, begins at the 32-ft-dia boring machine at the tunnel face, long before the muck reaches the conveyor. Directly behind the boring machine, which is averaging 7.5 ft per hour through the hard dolomite-limestone rock, muck is dumped on a 168-ft-long horizontal conveyor that rides 15 ft off the ground. The muck is loaded into 25-cu-yd cars that are hauled by locomotives then pulled by a cable system to the car-dump hopper. There the rock is dumped into a grizzly—a type of screening device where rocks larger than 14 in. are sifted out and passed through a crusher before being returned to the conveyor.



Tunnel network will control sewer overflows.

The muck travels from underneath the dump hopper on an inclined belt that feeds the vertical belt. At the top of the shaft the vertical conveyor belt extends upwards another 15 ft where the muck is transferred to a stacking pile.

Variation on a theme. A variation of the vertical conveyor system—an inclined conveyor belt—is being used by the joint venture of Kenny, Paschen, and S & M Contractors, Inc., Solon, Ohio, on another $63-million, 10-mile leg of the project. That portion of TARP was started in late 1975 with the tunnel being driven in two directions simultaneously using a 22-ft-dia boring machine in one direction and a 30-ft-dia machine in the other.

The joint venture decided to remove muck from a single shaft at mid-point in an effort to cut in half the necessary support crews and facilities for the tunnel job. The contractors had reservations about the possibility of being "muck-bound," that is, generating more spoil from two large tunneling operations than one shaft could handle. But the inclined belt erased those early worries. The contractors were concerned that if the belt broke, the entire mucking operation would be forced to shut down, but decided that the cost savings outweighed the potential downtime. "The whole tunneling operation depended on that belt. It would either make the job go or make it fail," says James C. Kenny, director of cost accounting for Kenny Construction.

In order to install the inclined belt, Kenny drove about 600 ft of 8-ft-dia tunnel on a 20-deg slope parallel to the main tunnel. A 300-ft trailing gear system directly behind the 30-ft-dia boring machine accommodated twin six-car mucking trains pulled by 25-ton locomotives. Once the 26-cu-yd muck cars are pulled up to the loading area behind the boring machine, an elevated conveyor dumps the muck into the cars using a chute to load one train and then move sideways to load the second train on the other side of the conveyor.

The trailing gear system brings the trains to a rotary dumper which dumps the muck cars at a rate of one per minute. A surge hopper beneath the rotary hopper collects four car loads of muck which is then fed to the belt by a vibrating feeder underneath the rotary hopper. A grizzly removes the large rocks, which go through a crusher and then back to the belt. A transfer belt moves the muck from the feeder to the main inclined belt.

The main 36-in.-wide conveyor belt runs 785 ft up the inclined tunnel and 110 ft above the shaft. The belt moves about 600 tons per hour of muck and is powered by five 100-hp motors coupled to five hydraulic pumps. Originally, four pumps powered the belt, but adding the fifth unit boosted the belt's hauling capacity by 20%.

One step further. The inclined belt was selected because at the time of the contract award in October, 1975, it was the fastest proven means for removing muck. "At the time we were not aware of the vertical belt, which takes this inclined system one step further," says Mixon. Adds Kenny, "Our inclined system worked beautifully, but the vertical system takes up less room, is an easier piece of machinery to move and is more cost effective."

The five remaining TARP projects under construction are using more conventional mechanical hoist systems. Two variations on the hoist are being used by the joint venture of J. F. Shea Co., Walnut, Calif., and Peter Kiewit Sons' Co., Omaha, on its $102-million, two-tunnel operation. The main 32-ft-dia tunnel is about 2.6-miles long and joined at mid-tunnel by a smaller, 15-ft-dia, 1.3-mile bore.

Due to the larger quantity of muck to be removed from the larger tunnel, slated for completion later this year, a hoist mounted on a headframe at the top of an 80-ft tower above the shaft lifts 25-cu-yd muck boxes through the 30-ft-dia, 260-ft-deep shaft using a counterweight system. This hoisting system is able to remove about 300 tons of muck per hour.

A similar system is being used on the $99-million, 4.5-mile tunnel being bored by the joint venture of S.A. Healy Co., McCook, Ill.; Gordon H. Ball, Inc., Danville, Calif., and Horn Construction Co., Inc., Merrick, N.Y.

However, the vertical conveyor belt outweighs all the other systems and could become the premier material removal system for tunneling projects, says William Paschen, project manager for Paschen Contractors. "Any tunneling operation is a materials handling operation, and if you can lick that problem, you've got it made. We decided to take a risk, and we won."

The controversial project, which is the largest project ever undertaken in the the U.S. for controlling overflows from combined storm and sewage-water systems, has two phases: a $1.9-billion pollution control stage, now about half contracted out, and a $900-million flood control stage.

TARP has been the target of a highly critical report by the General Accounting Office (GAO) that charged the giant project could cost up to $11 billion if the job is finished by its original 1986 deadline (ENR 5/24/79 p. 42). In a follow-up report in January, GAO rapped Chicago's deep tunnel plan and others like it proposed elsewhere as too costly a method for coping with combined sewer overflows. The Environmental Protection Agency estimates it will cost almost $26 billion to solve the polluting side of the combined overflow problem and controlling flooding could cost another $67 billion. ▫

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Air System Takes Rock to Surface

A Pennsylvania coal company uses compressed air to transport rock from low-coal mines to the surface. This pneumatic system has aided in construction work.

David Brezovec, associate editor

Rochester & Pittsburgh Coal Co. (R&P) of Indiana, Pa., plans to transport rock from its underground mines to the surface with low pressure, high volume pneumatic systems. From July to December, 1980, a pneumatic system in R&P's Urling No. 1 mine, near Shelocta, Pa., transported rock from a 1,200-ft tunnel driven to connect the Urling No. 1 and Urling No. 3 mines, both of which operate in the 52-in.-high Lower Freeport seam.

The system pushed 16,500 tons of rock 380 ft to the surface through an 8-in.-dia pipe. The system will stay at the Urling mines and pneumatic systems will be introduced into other R&P mines to eliminate underground gobbing and hoisting of rock.

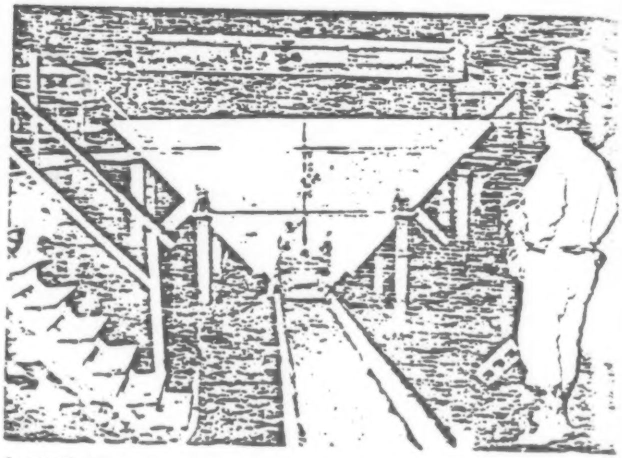
Handling rock is troublesome in thin seams

Eugene Jones, vice president of operations for R&P, explains that continuous miners at most of the company's mines, which are located in thin western Pennsylvania coal seams, have to take up as much as 2 ft of bottom clay in belt headings. Gobbing this rock congests air courses, interferes with room development and takes workers off production crews. Using the hoist to take rock to the surface can interfere with supply deliveries.

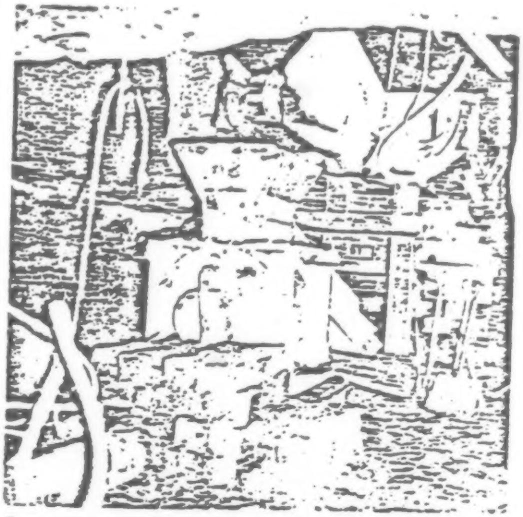
Previously R&P transported rock to the surface on the belt during weekend shifts. This system was inefficient because clay will stick to the belt, wearing it out and making it less suitable for coal haulage. Also, beltlines from many of R&P's Keystone Division mines go directly to the mine-mouth Keystone Electric Generating Station, operated by Pennsylvania Electric Co., and is used without passing through a cleaning plant. Rock taken up on the belt must be diverted to a rock dump.

Now, R&P will transport clay, shale and sandstone to the surface through the pneumatic system. The system also can be used to backfill cavities caused by roof falls. The pneumatic system, supplied and installed at a cost of about $150,000, consists of a Radmark airlock feeder, a positive displacement blower, a console with controls for the feeder and blower, and pipe and elbow joints.

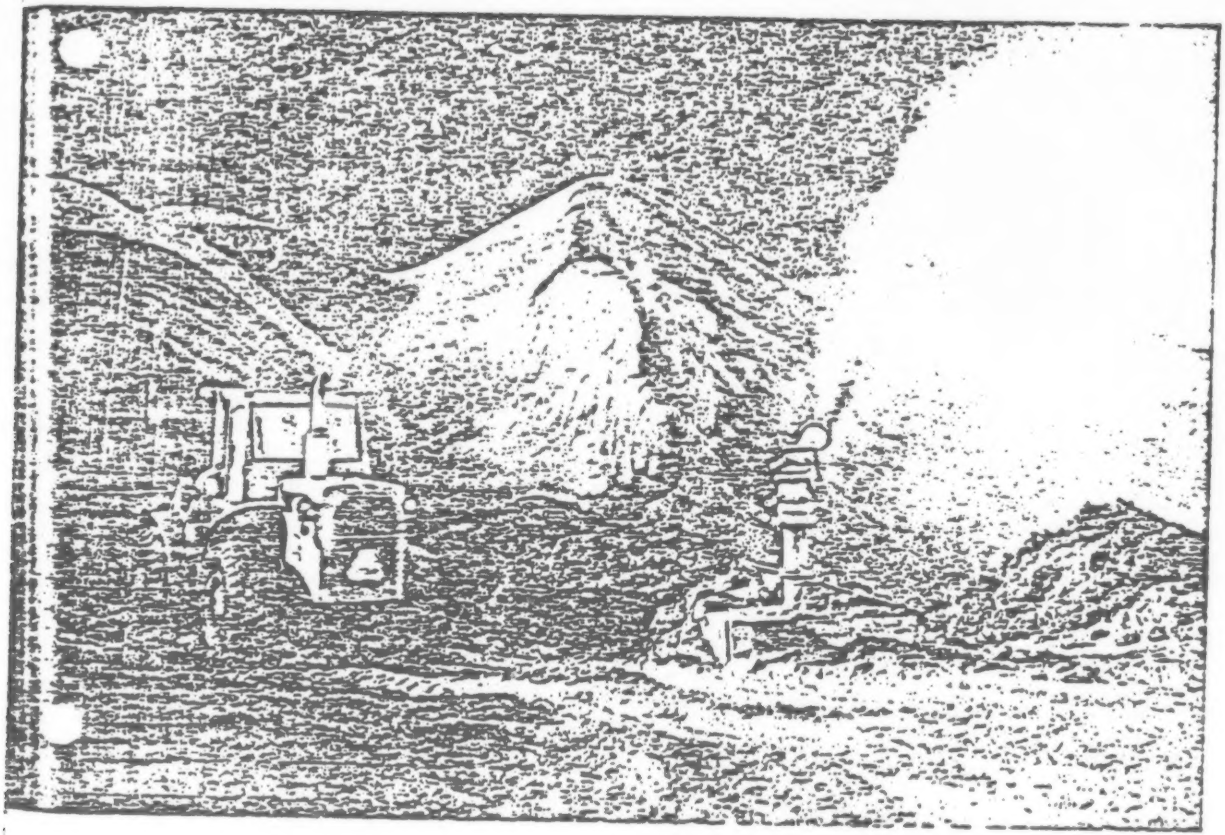
Rock is dumped into an 8-ton-capacity steel-reinforced ratio feeder hopper and is gravity fed onto a chain conveyor. A control gate maintains the flow into the pneumatic feeder



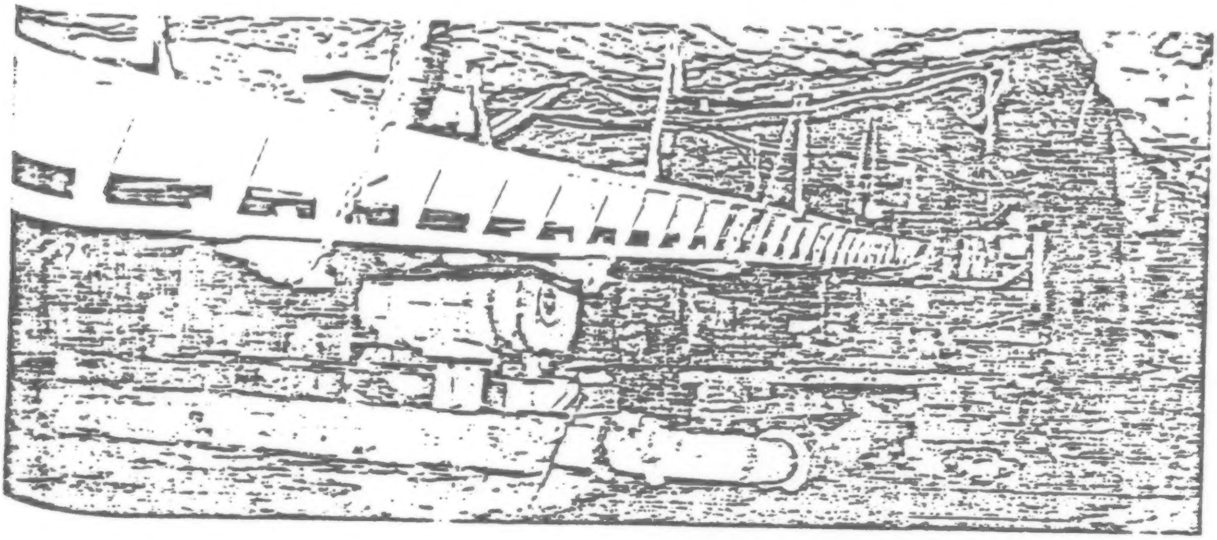
A control gate at the base of the 8-ton-capacity feeder hopper keeps the coal flowing onto the conveyor at 50 tph.



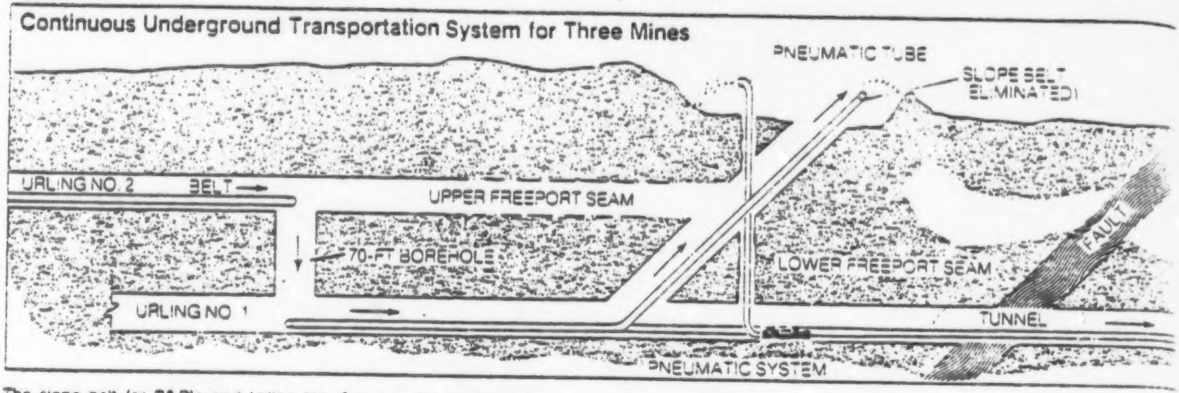
Coal from the conveyor falls into an airlock feeder that feeds it into the compressed-air chamber



On the surface, the mine rock is sprayed into a pile. It is loaded onto trucks by a front-end loader and used for road construction.



A positive displacement blower pushes air through the airlock chamber where it picks up rock and pushes it to the surface.


Continuous Underground Transportation System for Three Mines
PNEUMATIC TUBE
SLOPE BELT (ELIMINATED)
URLING NO. 2
BELT
UPPER FREEPORT SEAM
70-FT BOREHOLE
LOWER FREEPORT SEAM
FAULT
URLING NO. 1
TUNNEL
PNEUMATIC SYSTEM

The slope belt for R&P's and Urling No. 2 mine will be eliminated when the borehole to Urling No. 1 is completed.

at 50 tph—the capacity of the pneumatic system. The system will accept minus-2-in. material. A metal grate on the feeder prevents large lumps from flowing into the system. A 960-rpm motor sends a 12-psi stream of air from the blower at 120 fps. The console operator controls the feeder and regulates air pressure to send the rock to the surface through a vertical borehole.

The rock handled by the pneumatic system from the tunnel construction project was already reduced in size by the tunnel boring machine used to drive the tunnel. A crusher will have to be incorporated into the system before it can handle mine rock cut by continuous miners.

For the tunnel construction project, the pneumatic system was located in a 10-ft-high space, 12,000 ft from where the tunnel boring machine began driving into the fault that separated Urling No. 1 mine from Urling No. 3. The tunnel boring machine fed rock into 8-ton-capacity side-dump mine cars. Two sets of five of these cars traveled on track laid between the tunnel borer and the feeder hopper.

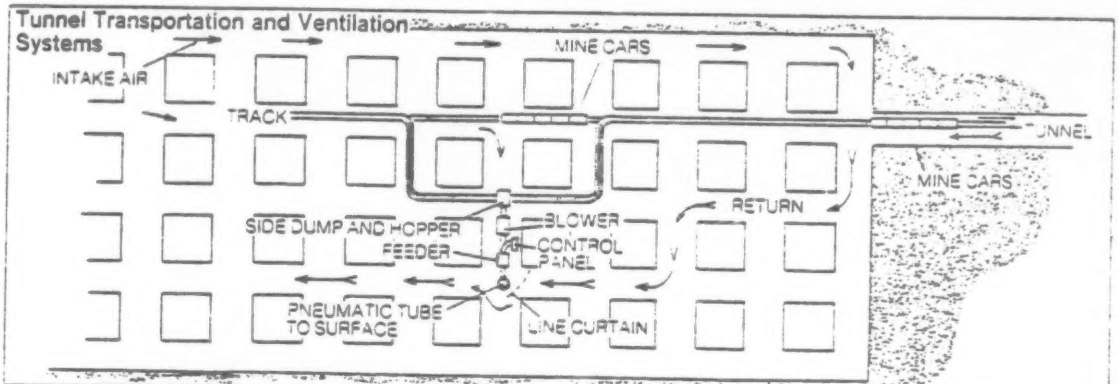
Controlling the dust

Rock travels through the pneumatic system to the surface at 100 fps. During the tunnel construction, the rock was shot straight from the open end of the pipeline into the side of a hill. Water was sprayed into the airlock feeder to control dust, but often the rock dried as it was forced through the compressed air system and a cloud of dust arose as the refuse shot out. If the outlet is ever located near farmland or homes, the rock will have to be shot into a bunker and filtered through a cyclone or bag filter.

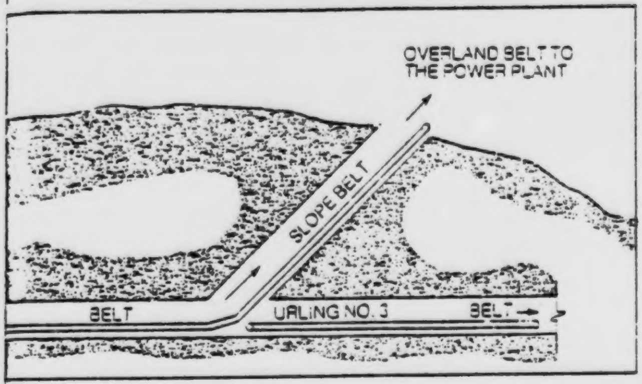
Haydar Gezik, chief mining engineer for R&P's Keystone Division, which operates the Urling mines, says the pneumatic system caused no unexpected problems. Occasionally, the pipeline would clog up with clay, and water was pushed through the system to clean it. Also, the 90-deg elbow joints sometimes wore out, but this was expected and the joints were segmented so that each segment could be replaced as it wore out.

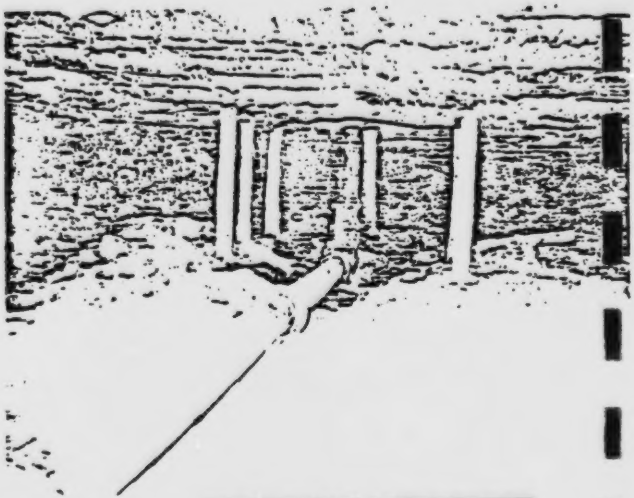
Low pressure keeps the system safe

Because the pneumatic system is a low pressure, high volume air system (without material in the system, air pressure is 6.5 lb at volumes of 5,000 to 6,000 cfm), it is safe in the mine environment, even when leaks develop.


Tunnel Transportation and Ventilation Systems
INTAKE AIR
MINE CARS
TRACK
TUNNEL
MINE CARS
RETURN
SIDE DUMP AND HOPPER
BLOWER
CONTROL PANEL
FEEDER
PNEUMATIC TUBE TO SURFACE
LINE CURTAIN

Tunnel rock is loaded onto mine cars that run on track laid between the tunnel boring machine and the pneumatic system.


OVERLAND BELT TO
THE POWER PLANT
SLOPE BELT
BELT
URLING NO. 3
BELT



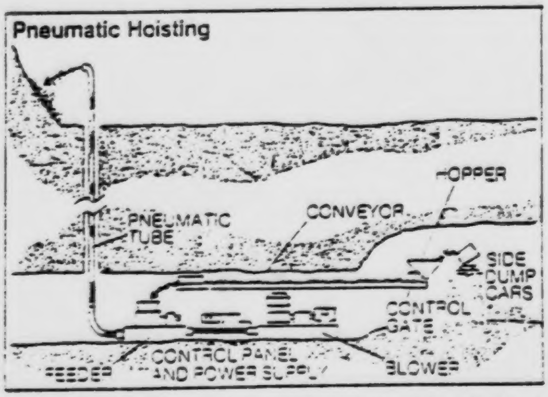
The rock, pushed by air, makes a 90-deg turn in the pneumatic tube on its journey to the surface. The 90-deg elbow joints are segmented. Sections can be replaced as they wear out.

Pneumatic hoisting systems, like the one being used to handle rock at R&P, are used in some European coal mines to transport coal. These systems feature capacities in the 80-tph range.

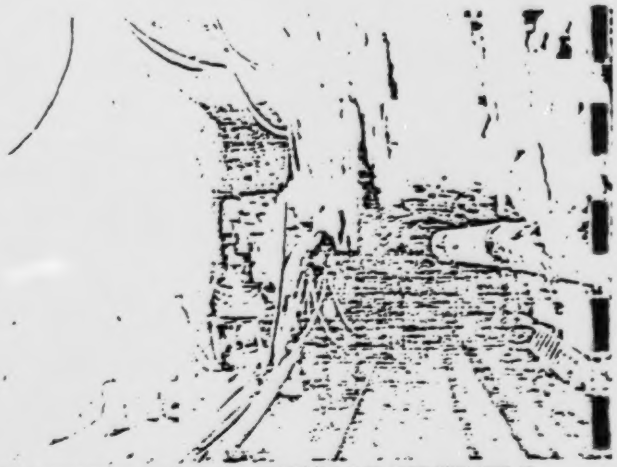
Coal at R&P's Urling No. 1, Urling No. 2 and Urling No. 3 mines is hauled to the surface on belts. The Urling No. 1 and Urling No. 3 mines are now connected by the tunnel and use the same 42-in. slope belt to the power plant. Urling No. 2 mine operates in the 42-in.-high Upper Freeport seam, directly above Urling No. 1. A borehole is being blasted from Urling No. 2 to Urling No. 1. When this project is completed, the three mines will share a common transportation system to the power plant. Coal mined at Urling No. 2 will drop to Urling No. 1's belt. Coal from both mines will travel on a belt through the recently opened tunnel into the Urling No. 3 mine. The coal from all three mines will then travel to the preparation plant on the Urling No. 3 slope belt.

Five miles in 50 minutes

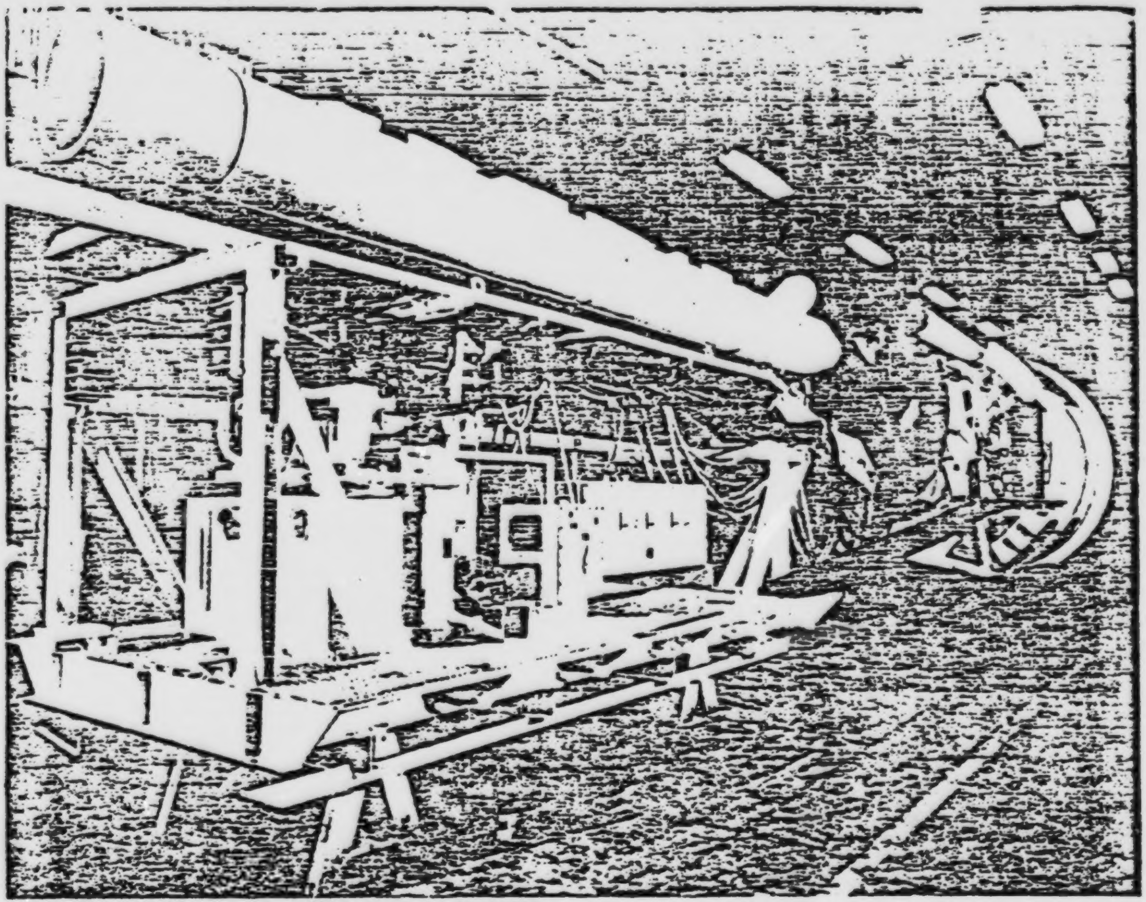
Approximately 7,000 tpd will arrive at the power plant through this belt system. Coal starting at the farthest point from the plant, in sections at Urling No. 2, will make the


Pneumatic Hoisting
HOPPER
PNEUMATIC TUBE
CONVEYOR
SIDE DUMP CARS
CONTROL GATE
CONTROL PANEL AND POWER SUPPLY
FEEDER
BLOWER

The pneumatic hoisting system consists of an airlock feeder, a blower, a control console and pneumatic tubing.



The tunnel boring machine was assembled in an excavated area at the fault.



A Jarva Mark 12-1400 drove the 12,000-ft tunnel. Air was pulled from the tunnel face through ventilation tubing.

5-mile trip to the plant in about 50 minutes.

R&P opened the tunnel between the Urling No. 1 and Urling No. 3 mines with a Jarva Mark 12-1400 tunnel boring machine. The machine, used in the construction industry for hard rock tunneling, was modified for permissibility in the underground coal environment with fully enclosed motors and electrical cabinets.

The machine, operated by the Williamson Shaft Construction Co., cut a 14-ft-dia hole through sandstone, shale and coal at 12,000 psi. As rotary torque and forward thrust were applied to the full face cutting wheel, disc-type cutters penetrated the rock in a circular kerf pattern. This caused the rock to spall and break out of the tunnel face. The material was conveyed automatically to a muck handling system at the rear of the machine and dumped onto the mine cars.

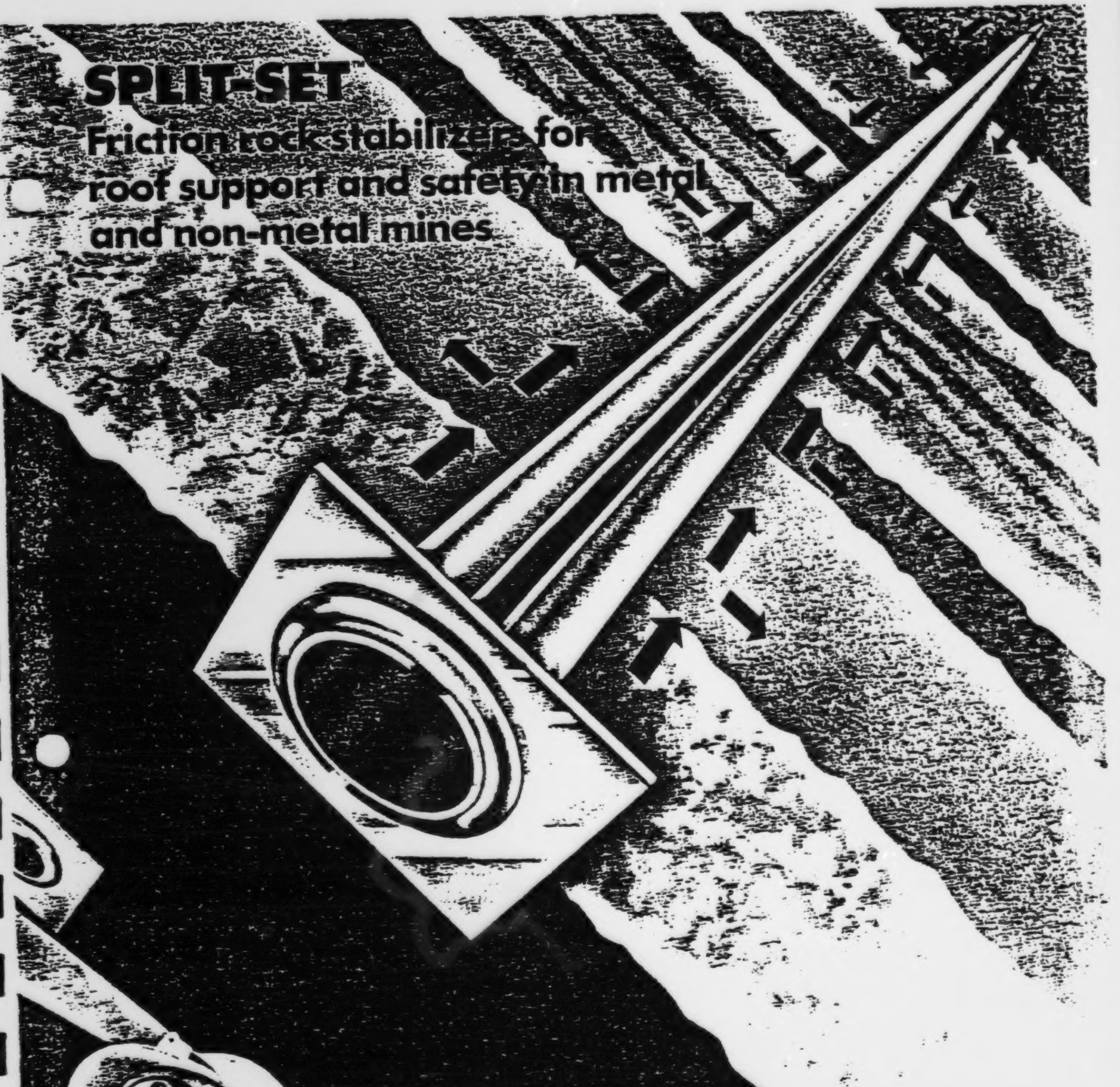
Exhaust fan ventilates the face

Water sprays and ventilation were used to suppress dust. A dust shield was mounted 4 ft from the cutting face and ventilation tubing was extended from the shield to a 30-in. exhaust fan that pulled air through the tunnel section. The tunnel and pneumatic system location were ventilated in the same manner as a regular mining section but without the high volume that characterizes coal mine ventilation.

Three headings were driven to the fault where the tunnel boring machine began working. Two of these headings were used for intake and one for return air. A separate split of air from one of the intake headings was directed into the area where the pneumatic system was in operation. The other went to the face of the tunnel, across the face, through the dust shield and out of the section through the ventilation tubing.

The 6-ft-dia borehole that will connect the Urling No. 2 and Urling No. 1 mines through 70 ft of rock is being drilled and blasted. A 10-in.-dia pilot hole was drilled from the surface through both mines. An air drill widened this hole between the mines to 24 in. in diameter. The area around this borehole is drilled and blasted using 4-ft blast holes. Material falls into Urling No. 1 and is gobbed in a nearby room. Soon the pneumatic system will be called in to transport this rock to the surface. ▬

SPLIT-SET™
Friction rock stabilizers for roof support and safety in metal and non-metal mines



The simple new roof support that holds firm in hard or soft rock and never needs torquing.

Ingersoll-Rand.

The SPLIT-SET support system offers a combination of benefits never available before.

- **Safer, more effective support.**
- **Reduces roof support costs.**
- **Maintains roof plate preload.**
- **Absorbs roof movement without loss of support.**
- **Anchorage increases with time.**
- **Doesn't loosen from adjacent blasting.**

Fig. 1 Loose-hanging roof plates like this can be a thing of the past when you use SPLIT-SET roof stabilizers.

When you walk down your heading, look up. How many of your roof-bolt plates are hanging loose? How much time do you spend tightening bolts, over and over again? Haven't you wished that some day, someone would invent a roof bolt that would really stay put?

The time has come, the product is here, and this is it!

How the System Works

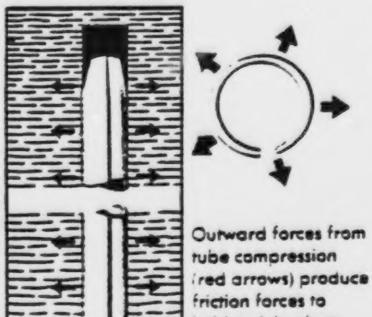
The patented SPLIT-SET stabilizer is one of the simplest and safest of all roof support systems. It's a hollow, compressible, high-strength steel tube with a slit extending its full length, a taper at the upper end and a retaining ring at the bottom.

All you do is drive the SPLIT-SET tube into a bore hole slightly smaller than the tube. Once it's in, it anchors itself by the outward expansion force of the tube against the rock, as shown in Fig. 2. This anchoring force is distributed throughout the entire length of the tube. Its holding power is at least equal to or greater than the force required to drive it—normally 0.75 to 1.5 tons per foot of tube length. Standard lengths are 4, 5 and 6 ft. (1.22, 1.52 and 1.83 m), with other lengths available.

That's all there is to it. No screw threads to tighten up. No separate nuts or shells to get lost on the job. No resins to mess with.

Greater Safety

The safeness of a roof support device is measured by its ability to support the mine roof, and maintain this support under mining conditions. Tests show that the SPLIT-SET rock stabilizer holds firm under conditions that cause loosening or failure of other roof bolts. Tests have also shown that roof movement and shock waves from nearby blasting actually tend to anchor the SPLIT-SET support more firmly in the hole and the longer it stays in the rock, the tighter it holds.



Outward forces from tube compression (red arrows) produce friction forces to hold rock in place.

Fig. 2 SPLIT-SET stabilizer holding forces.

Saves Time and Money, Too

The simplicity and holding power of SPLIT-SET stabilizers save time and money on every hole. Every one you install is a *working* roof support. Once it's in, with its support plate tight against the rock, you're finished with it. No man-hours spent tightening bolts or checking torque, initially or to correct for loosening later. And no time lost to insert and break resin bags, rotate bolts to mix and hold for set-up, and no lost materials from accidental resin bag breakage.

Usually the same equipment used to drill the bore holes can be used to set the stabilizers. I-R Jackdrill® rock drills, drifters or stopers provide the impact force and feed to drive the split tube into the slightly smaller bore hole, much as metal is forced through a die.

Although effective in any type of rock, SPLIT-SET roof supports are particularly effective in soft or unconsolidated formations, where timber sets have been used before. In one mine, SPLIT-SET supports saved $100 per foot of drift compared to timbering. Under these conditions you can cut roof support costs in half. In other cases they simplify

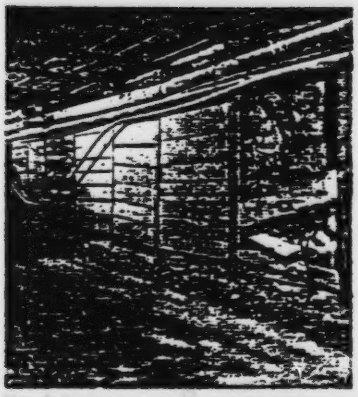


Fig. 3 Timbering provides acceptable support for unconsolidated rock.

Fig. 4 But SPLIT-SET stabilizers, with mesh, do the same job at about half the cost.

installation of roof mesh which goes up, with the tube and roof plate, in a smooth, single operation. (See Fig. 3 and 4.)

Maintains Roof-plate Preload

With SPLIT-SET stabilizers a tight roof plate *stays* tight. After installation, the compressed high-strength steel tube maintains its outward holding pressure on the rock—a pressure acting on every inch of its length. There's none of the stress concentration that so often causes point anchor bolts to crush the rock at the contact points and lose holding power. Loose-hanging roof plates often result. (Fig. 1.)

Absorbs Roof Movement Without Loss of Support

A SPLIT-SET support is the only practical roof stabilizer that can absorb substantial amounts of roof movement without loss of holding power. That's because the holding force consists solely of pressure and friction, acting throughout its length. When downward force of rock movement exceeds the holding force, the SPLIT-SET tube simply slips momentarily, then normally re-anchors itself immediately with the same force as before. Under similar conditions, a point anchor bolt or resin bolt may fail, leaving the roof without the proper support.

SPLIT-SET stabilizers can also withstand considerably greater shearing stresses from lateral shifting of the bedding planes. Because it is a tube instead of a solid rod, it can be partially collapsed and deformed at a slip plane under strains that could cause shear failure of other rods. This deformation, illustrated in Fig. 5, anchors SPLIT-SET supports more firmly in the rock.

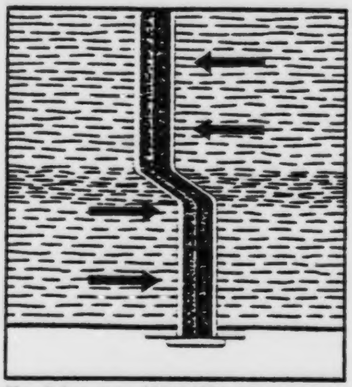


Fig. 5 Reaction of SPLIT-SET tubes to horizontal shifting of bedding planes.

Anchorage Increases with Time

Tests have shown that the holding power becomes stronger as time goes on. Why? Because rock movement, blasting shocks and surface bonding at the bore hole all contribute to a cumulative increase of frictional resistance between the tube and rock. Pullout strengths have been observed to more than double after only a few weeks.

Installation

SPLIT-SET stabilizers need no special equipment for installation. They can be driven into the roof or rib with the same rock drills used to make the bore holes, as in Fig. 6. I-R Jackdrill® rock drills or drifters with a 3-inch (76 mm) or larger bore will have ample power. A simple adapter for the drill is the only accessory required. This adapter, available from Ingersoll-Rand, fits into the drill chuck, and accepts the ring-end of the SPLIT-SET tube. The roof plate, held between the retaining ring and the roof, goes up with the tube. Detailed installation instructions are supplied with your SPLIT-SET order.

Applications

SPLIT-SET stabilizers have been used in quantity in lead, zinc, copper, gold, trona, limestone, fluorspar and uranium mines with outstanding success.

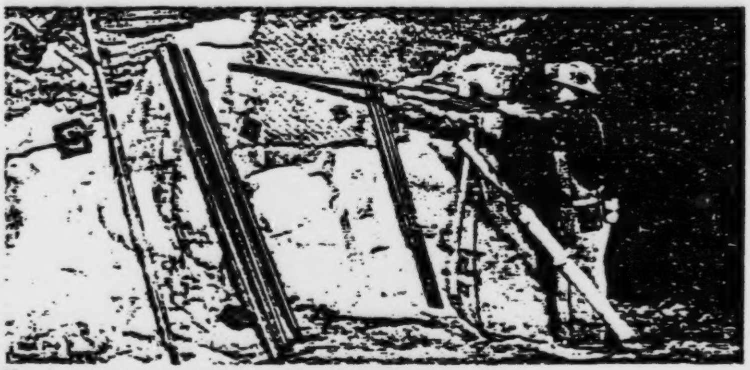


Fig. 6 Jackdrill with adapter installing SPLIT-SET rock stabilizers.

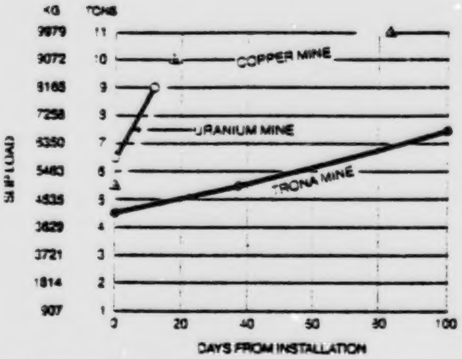


Fig. 7 Typical slip load increase with time for SPLIT-SET stabilizers.

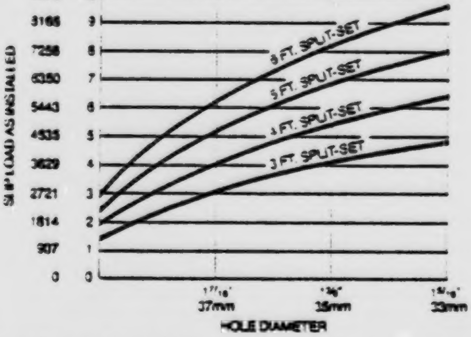


Fig. 8 Typical slip load for SPLIT-SET stabilizers of various lengths in various hole sizes.

The uranium mine applications provide unique examples of the ability to customize the installation to meet geological conditions. The roofs in many of these mines consist of very soft and friable sandstones, mudstones, or shales. Here SPLIT-SET stabilizers were installed in holes smaller than those normally used for hard rock, to provide the required stabilization and holding power. Shock waves from adjacent blasting that tore loose other bolts served to anchor the SPLIT-SET tubes more firmly in the rock. Similar performance advantages have been obtained in hard rock installations as well.

Operating Results

The unique features of the SPLIT-SET stabilizer have permitted substantial improvement in roof support and safety under many underground conditions.

The curves in Fig. 7 show the increase in slip load with elapsed time after installation of SPLIT-SET stabilizers. These curves are based on test results at three mines, typical of many others. Note that the holding load actually increases within a few weeks to 150 to 200% of the value at initial installation. This sharply contrasts with many mechanical bolt installations which often require re-torquing after a few weeks.

Fig. 8 shows typical test results for SPLIT-SET stabilizer holding load in holes of different diameters with tubes of different lengths, in hard rock. Experiments in your own mine will determine the best hole size and tube length for the holding power and installation speed you wish to maintain.

Application Engineering

Because holding power varies widely with roof geology, prospective users are urged to take advantage of our free application engineering service. This will assure the correct tube length, hole size and drilling adapter for specific installation conditions.

Manufacture and Distribution

SPLIT-SET stabilizers are manufactured under close quality control to provide consistent, uniform holding power per unit of tube length. We anticipate a substantial and rapid increase in demand for SPLIT-SET stabilizers as their unique benefits become more widely known, so we have ample production capacity to meet all needs for the metal and non-metal mining industry.

SPLIT-SET Accessories

The following accessories required for SPLIT-SET stabilizer installation, should be specified on your order.

Drill Adapter

The simple adapter shown in Fig. 9, enables your drilling equipment to hold and drive SPLIT-SET stabilizers with no other modifications.

Specify the make and model of drill to be used, the steel size and chuck length.

Roof Plates

SPLIT-SET stabilizers require a roof plate with a 1-9/16" (40 mm) diameter hole. These plates are available from I-R in the standard size 6" x 6" x 1/4" (150 x 150 x 6 mm). Note that the hole diameter is larger than used in plates for most other roof bolts.

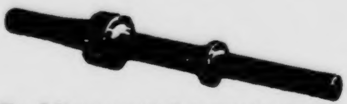


Fig. 9 Simple Jackdrill adapter for SPLIT-SET stabilizers.

FOR ADDITIONAL INFORMATION ...

In U.S., contact IREC Sales Agent, Scott MTS Inc., LeComa Star Route, Rolla, Missouri 65401 (314) 364-4008.

In Canada, contact Canadian Ingersoll-Rand Company Ltd., P.O. Box 610, Station B H3B 1M2, Montreal, Quebec, Canada (514) 395-7321, or your local Ingersoll-Rand Office.

Ingersoll-Rand Equipment Corporation SPLIT-SET DIVISION
P.O. Box 301, Princeton, N.J. 08540 (609) 921-9103

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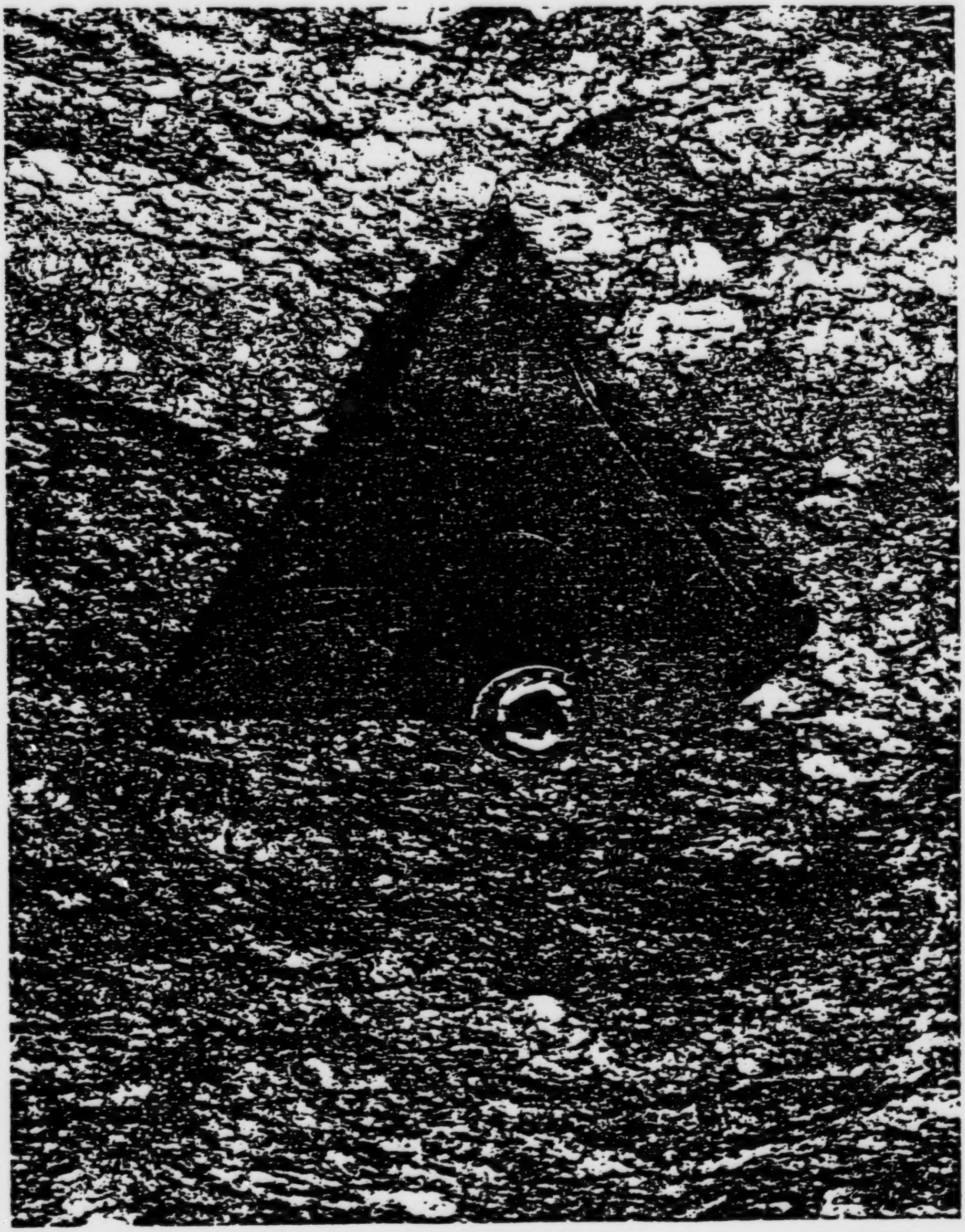
Rock reinforcement system

SWELLEX®

for high speed rock bolting

Atlas Copco



SWELLEX® for immediate reinforcement within seconds

For many years Atlas Copco has conducted extensive research in high pressure water technique and rock reinforcement. This has resulted in SWELLEX, a simple method of rock bolting using a reshaped steel tube.

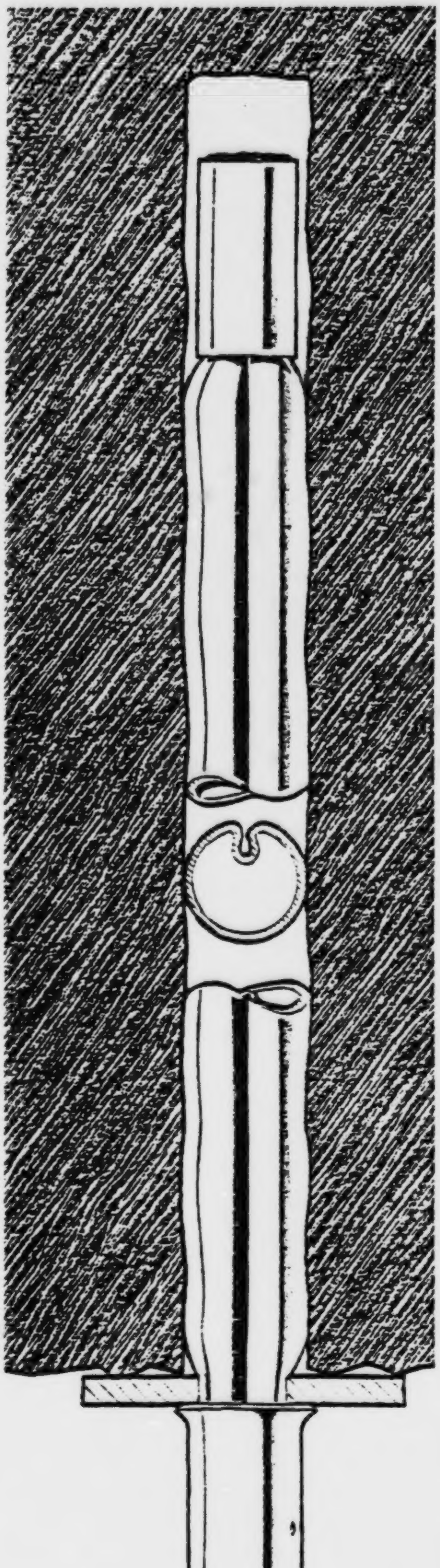
The SWELLEX system consists of the "bolt", an installation rod with chuck and a high pressure water pump.

The bolt is made of a steel tube, the outer diameter of which as been reshaped to assume a smaller external diameter. The ends of the tube are reinforced with short sleeves sealed through welding.

When setting a SWELLEX bolt, water at high pressure is pumped into the tube through a connection at the lower sleeve. This causes the tube to swell in the hole, close to its original size.

Major benefits and design features of the SWELLEX® rock reinforcement system

- Very fast to install, about 50 bolts per hour Shorter bolting times leaves more time for production
- SWELLEX bolts give immediate support
- Bolts can be set in holes varying in diameter from 30 to 39 mm. Thus SWELLEX is applicable in gauge wearing rock.
- Good resistance against vibration. Can be used quite close to the face.
- Can resist large shear deformations
- Good properties in blasted rock. Do not get caught up in crushers etc.
- A rod is used for installation. The miner doesn't have to stand under the bolt when installing it. No platform is needed for roof heights up to 4 m.
- No rotation, high torque or high feed force required while bolting, only a high pressure hose attached to the bolt.
- Only water is used, no pollution from chemicals or cement.



How it works

SWELLEX is manufactured from Ø 41 mm steel tubes, which have been mechanical reshaped to assume an outer diameter of only 28 mm. Sleeves are pressed onto the ends and sealed through welding. One of the sleeves retains the washer in place.

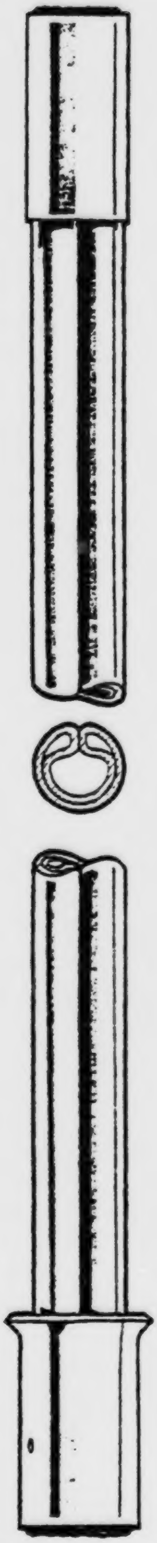
A small hole is drilled through this sleeve, to allow water to be injected into the bolt during installation.

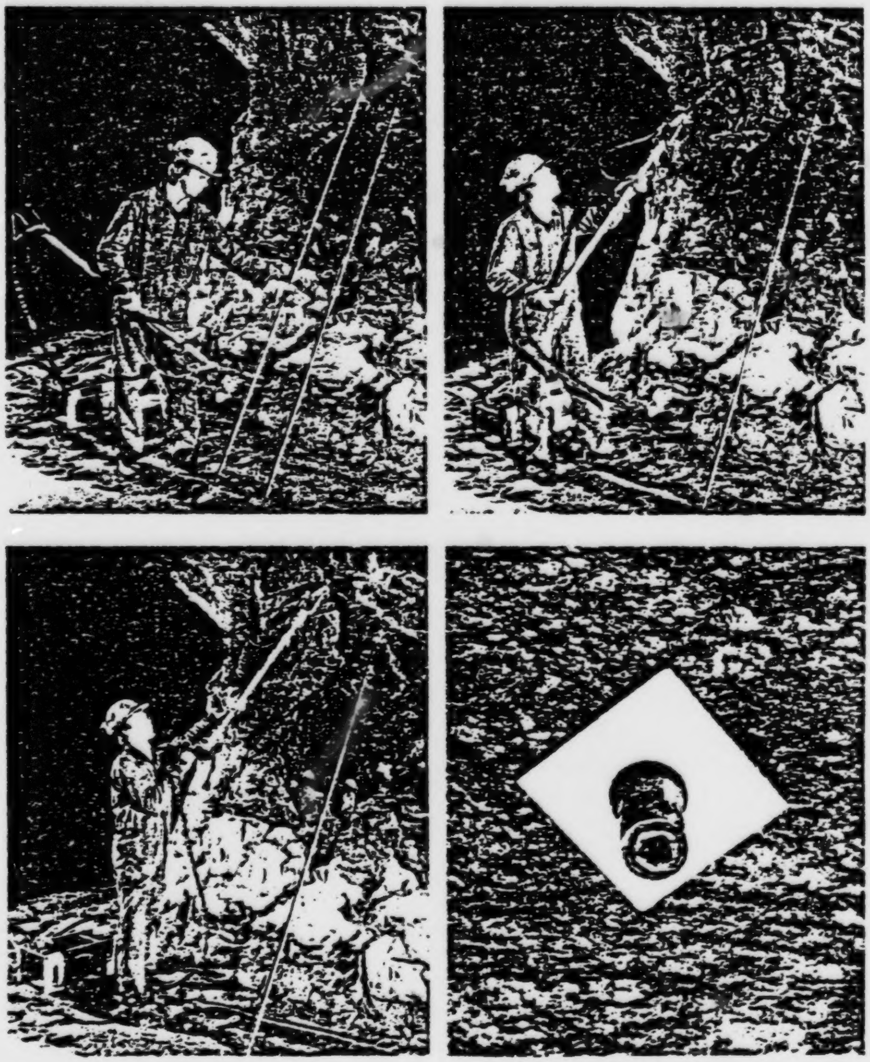
During installation, the high pressure water makes the bolt expand in the drill hole. As the drill hole diameter is smaller than the original diameter of the tube, a tongue is left inside the profile.

This tongue is compressed by the water pressure. And, when the pressure is released, the tongue acts as a spring producing radial forces on the bolt, so that the bolt continuous to press against the walls of the hole.

Through this swelling process, the length of the bolt is reduced through contraction, resulting in the washer at the lower end tensioning firmly against the rock.

Due to the high water pressure, the bolt will follow irregularities in the drill hole and hence increase frictional forces.





Technical data

Fracture load	100 kN
Friction force	100—200 kN/m
Water pressure	300 bar
Drill hole diameters	30—39 mm
Bolt lengths	1.2—3.0 m in intervals of 0.3 m other length on request.

High pressure water pump

Pneumatic water booster pump suitable for mobile use. The pump is connected to the ordinary water and air systems at the jobsite. A separate water reservoir can also be used.

The pump is available in flame-proof version.

Atlas Copco

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Cementitious rock bolt cartridge for anchors

One of the most exciting developments to take place for many years in the field of rockbolting is the international launch of a new cement anchor by Commercial Plastics Special Products, a member of Nairn International. Known as Cemicron 2000, the system utilises micro-encapsulated water which can be mixed dry with a variety of cement products and used in cartridge form to anchor roof and rock bolts. Water mixes with the cement only when the bolt is inserted into the cartridge.

The patented process was invented by Robert Simpson, a US Bureau of Mines engineer, and developed by the US South West Research Institute of San Antonio, Texas. The invention has already received acclaim from many quarters, and last year won the coveted IR 100 Award for one of the 100 most significant technical advances awarded by the US journal *Industrial Research & Development*.

Cheaper than resin bolts

The invention has resulted in a full column cementitious anchor for underground roof bolts and wood dowels which will sell at a price expected to be some 20 per cent lower than the current price of resin roof bolts. Extensive tests carried out over a period of four years in the US have indicated that the product will provide consistent anchorage of high strength at high setting speeds, more safely and more cheaply, as well as more conveniently, than alternative anchorage systems currently available.

Invention of the two-component polyester resin cartridge provided the first method of full column anchorage and avoided the major disadvantages of the mechanical roof bolt systems. During the '60s and '70s roof bolting increased and new techniques such as longwall face reinforcement with cuttable wood dowels were developed.

Although polyester resin cartridges have worked very well in roof bolting applications over the years, they suffer from two disadvantages. First, they are flammable and therefore represent a significant additional hazard underground. And secondly, because their principal raw materials are oil based, costs have had to increase in line with oil price increases.

No flammability hazard

It is for these reasons that throughout the world mining authorities such as the US Bureau of Mines have been seeking safer and cheaper alternatives to the resin cartridge. Cemicron 2000 provides all the convenience of the resin cartridge without any flammability hazard, and costs less.

It is the first anchoring system to have the precise amount of water in a micro-encapsulated form already mixed with quick setting, high-strength cement. The micro-encapsulated water activates the cement and enables exact metering of water within the cartridge. The wax shells of the tiny, spherical microcapsules are strong enough to resist fracture during normal handling, but collapse under the pressure caused by insertion of a steel bolt or wood dowel.

Pressure build-up causes the crushing of microcapsules in advance of the bolt tip, releasing water uniformly across the hole. The homogeneous grout then formed hardens rapidly to provide firm, full-length anchoring. Even in vertical roof bolting applications, near-immediate bolt holding is achieved so that the hand-drill or bolter can be withdrawn straightaway. Full-load setting is achieved within a few minutes.

The bolt or dowel has no mixing to do because the micro-encapsulated water is already intimately mixed with the cement. There is no need for immersion in water as with some other cementitious systems—an obvious advantage in terms of convenience.

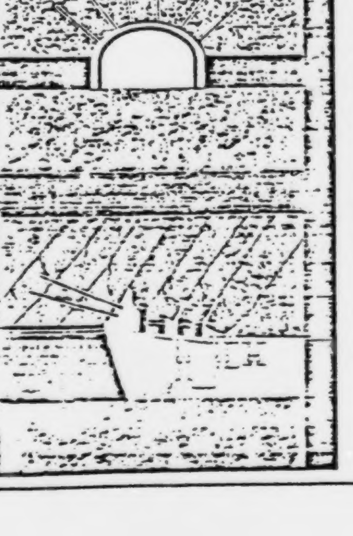
The speed of setting of Cemicron 2000 is easily adjusted to accommodate the required application at no extra cost. Firm anchorage can be achieved in under half a minute if required. However, a range of setting times will be available.

The system is absolutely safe for underground mining and tunnelling, is non-flammable, non-toxic and non-allergenic. Cartridges will be made in different diameters and different lengths to suit the expected markets. In the UK, the market is accessed at up to 5 million/year. The National Coal Board is most unlikely to move away from steel arches as its prime method of strata control, but roofbolting will develop as an important secondary method in UK coal mines.

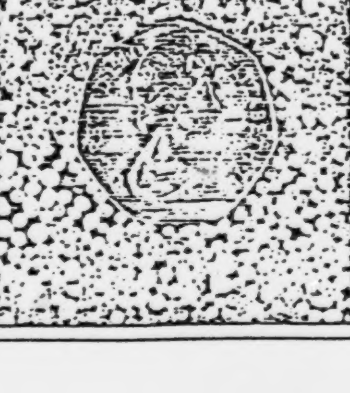
Export potential

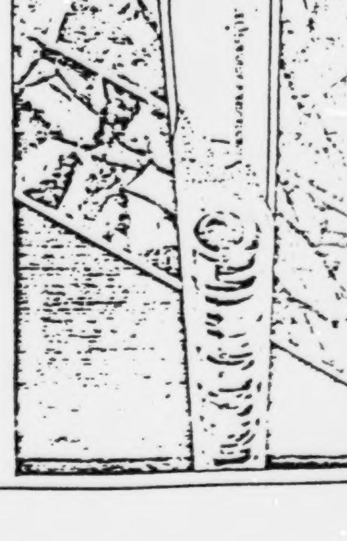
The product has a high sales potential in all mining countries, will be invaluable in tunnelling, and has civil engineering applications above ground too. Commerical Plastics will manufacture the product in Britain, and intends to sell directly in the UK, but is looking for suitable overseas agents in export markets. It has obtained exclusive licences for Canada, Australia and South Africa.

David Martin



(Left) Roof bolting applications using Cemicron 2000 cementitious cartridges. (Below) Micro-encapsulated water showing size relative to UK 50p coin. (Right) Microcapsules of water collapsing under pressure as the bolt is turned.





by G. B. SPARKS

Application of Hard Rock Continuous Miners to Cut-and-Fill Slot Stoping

The occurrence of economic deposits of the borate minerals colemanite and ulexite are relatively rare on a world-wide basis. There are deposits in Turkey, South America and the southwestern United States. In southwestern U.S. there are notable deposits worked by United States Borax and Chemical Co. at Boron, Calif. and extensive deposits located in and around Death Valley, Calif., including those at the Billie mine of American Borate Co. in the upper Furnace Creek Wash region within the confines of Death Valley National Monument and National Park Unit.

Borates have been mined sporadically in Death Valley since 1881 when Aaron Winters originally collected cotton ball ulexite from the floor of the valley just north of what is now known as Furnace Creek Ranch. Pacific Coast Borax, the predecessor to U.S. Borax, was active in the Death Valley area from the late 1800s until the mid 1920s. It was in this area that the famous Twenty Mule Team wagons operated. In 1971 Tenneco Oil Co., American Borate's predecessor in interest, resumed colemanite and ulexite mining in Death Valley at the Boraxo open pit located in upper Furnace Creek Wash. This pit was operated from 1971 to 1977 at which time the reserves found along one limb of a large anticline were depleted.

In late 1976, in response to increased environmental pressures to reduce or stop mining within Death Valley National Monument, the Congress of the United States passed Public Law 94–429. This law imposed a four-year moratorium on further surface disturbance for the purposes of mineral extraction. There were some minor technical exceptions; however, it greatly increased the difficulty in meeting environmental and other regulatory conditions involved with mining within the national park unit. The advent of this law along with other considerations caused American Borate Co. to proceed with the Billie mine as being the most environmentally compatible and most economically viable property currently held by American Borate.

Billie mine ore zone about 300 ft thick

The Billie borate ore deposit (named after the madame of a local house of ill-repute) is a strata-controlled lensatic body located in a calcareous shale country rock. The ore body itself varies in depth from 400 ft to in excess of 1200 ft and dips at approximately 17°. The body is relatively short in strike length compared to the length up and down dip. The average thickness of the ore zone is about 300 ft.

The ore is highly interbedded with calcareous shales and limestone. There has been relatively minor tectonic activity in the ore body with most faulting being post-depositional. Ground water has mobilized borates to remineralize these faults so that there are no large barren expanses contained within the body itself.

Rock strength, a critical factor in the use of continuous miners, ranges from a low of 2000 psi unconfined compressive strength for ulexite and softer shales to a high in excess of 20,000 psi unconfined compressive strength for probertite (a sodium calcium borate similar to ulexite). The abrasivity of the rock, also an important factor in the use of continuous miners, is very low, the greatest hardness being that of colemanite at about 4.5 on the Moh's scale.

The geographic location of this deposit is important to establishing the criteria by which the mine system was established. The deposit is located in the eastern portion of Death Valley National Monument approximately 140 miles west of Las Vegas, Nev. It is immediately adjacent to the park boundary in upper Furnace Creek Wash located between two mountain ranges—the Blacks to the west and the Green Waters to the east—under an alluvium filled valley, at about 2500 ft above sea level.

Mine targeted to produce 200,000 tpy

For sizing the operation initially, our customers' needs were indicated to be about 200,000 tons per year product combined colemanite and ulexite. The company determined there were sufficient reserves to produce at least at that rate for a number of years so 200,000 tpy was adopted as the initial production target.

The next step was the selection of the mining method (fig. 1). All the criteria influencing the mining method were listed and reviewed; they included the following:

- Ore had to be mined selectively because one product was sold run-of-mine grade with relatively tight grade specifications. Environmental restrictions prevented beneficiation at the mine site, necessitating a 40-mile one way haul to the nearest milling facility, so it was important to keep dilution to a minimum in order to hold trucking costs down.
- The mining method had to lend itself to mining a deposit with our depositional characteristics; that is, 300 ft thick, 17° dip and many interbeds of waste.
- The method had to be highly mechanized due to unavailability of skilled miners and the remote location (140 miles to the nearest town).

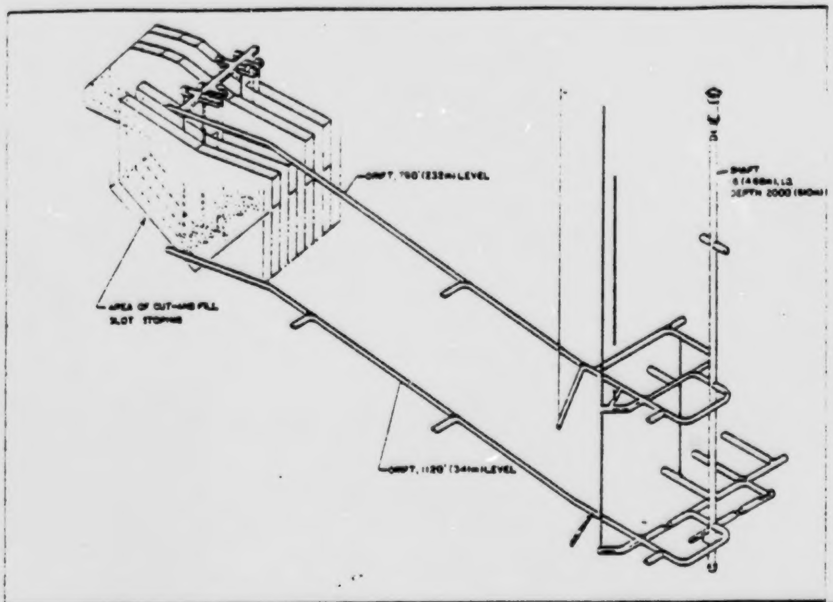
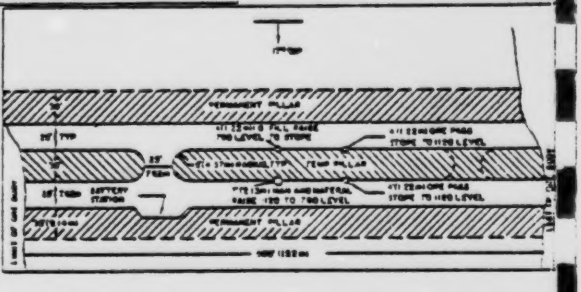


Fig. 1. Isometric development plan for Billie mine, American Borate Co.

Fig. 2. Stope layout of Billie mine


PERMANENT PILLAR
FILL RAISE 760 LEVEL TO STOPE
ORE PASS STOPE TO 1120 LEVEL
TEMP PILLAR
BATTERY STATION
ORE PASS STOPE TO 1120 LEVEL
PERMANENT PILLAR

- The mining method had to be environmentally compatible due to the location under a national park unit. No waste could be piled on the surface and no subsidence could be tolerated.
- The method had to be economically viable.

Even with this somewhat unique list of controlling criteria, the mining method selected may have been different had it been reviewed by other people under different circumstances. The author, having spent some time at the tiller of a full-face ripper in southern Illinois coal mines and having spent an equal amount of time pumping sand-fill in the stopes of the Coeur d'Alenes of Idaho, and our consultant, a progressive, widely experienced consultant, did not allow ourselves to be bound into conventions of either coal or hard-rock mining methods. Our review and decision making process went something like this:

> The thick ore body meant excavations of large vertical dimensions but mandatory surface stability precluded open stoping at this relatively shallow depth. Selectively mining alternating beds of ore and waste as thin as 2 ft severely strained the economics of the drill and blast method, even if we could have found the miners to do it.

Fig. 3. Checking bits on ripper head for cutting hard rock of up to 20,000 psi

Cut-and-fill stopes with tall narrow stopes (fig. 2) oriented along strike met the rock mechanics and waste disposal criteria. Continuous miners met the mechanization criteria, but full face machines are not selective enough and no continuous miners had routinely cut rock with compressive strengths in excess of 10,000 psi unconfined compressive strength in a production situation.

Two types of roadheaders purchased

It was at this point that we began our investigation into roadheaders. Small roadheaders have been popular for some time in Europe and had some application in U.S. coal mines mainly in the east to cut rock (that is as opposed to coal) for overcasts, etc. The rock was generally shale or soft sandstone and limestone of up to 5000 psi unconfined compressive strength. Almost always the application was in a development or maintenance capacity with a relatively low production requirement.

In fall of 1976, there were three manufacturers—one each Austrian, English and American—coming out with high production hard rock roadheader type continuous miners. The Austrian firm was in the earlier stages of development of their AM 100: therefore, we did not further consider that machine. The English company had developed an extremely large two-boom roadheader in response to requests by iron ore mining companies in

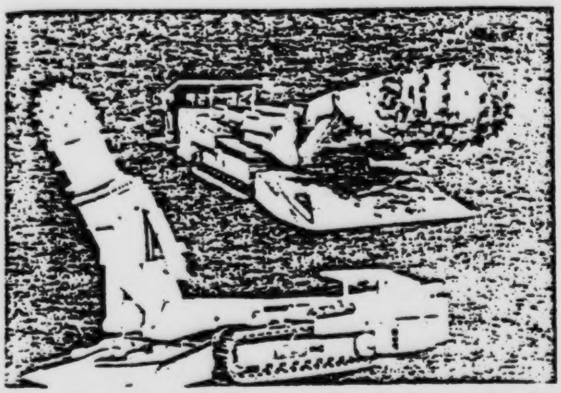


Fig. 4. Tests with identical cutter motors demonstrated that the ripper head had a production rate as much as one third higher than the auger head working under identical conditions

France. The American company had developed a medium-size single boom roadheader in an attempt to expand the role of continuous miners in coal development and medium hard rock production and development. There are, at the time of this writing, several heavy weight high horsepower machines made in Europe; however, ABC chose to further investigate only the English and the American firms due to machine design and the level of representation of the companies in the U.S.

The specific machines which American Borate invesitgated, assisted in the development of and subsequently purchased are as follows:

Super Roc-Miner 330

Manufacturer - AEC Inc., State College, Pa.
Machine weight - 45 tons (41,000 kg)
Cutter type - ripper and alternative milling (auger) head
Cutter horsepower - 215 hp (160 kw)
Hydraulic pump horsepower - 165 hp (123 kw)
Type traction - hydraulic
Gathering and conveying system - 60 hp (45 kw)
Electric drive to gathering arms and single-strand chain flight conveyor
Conveyor - slewing tail, foot shaft driven
Motor cooling - closed circuit, water

Dosco TB 600

Manufacturer - Dosco Overseas Engineering Ltd., Tuxford, U.K.
Machine weight - 90 tons (81,300 kg)
Cutter head - twin 285 hp (212 kw) milling (auger) type
Hydraulic pump - 200 hp (150 kw)
Type traction - hydraulic
Conveyor system - 60 hp (45 kw) tail shaft driven, fixed position, round link double-strand chain conveyor
Colling system - open circuit
Cooling water pumped through dust control sprays

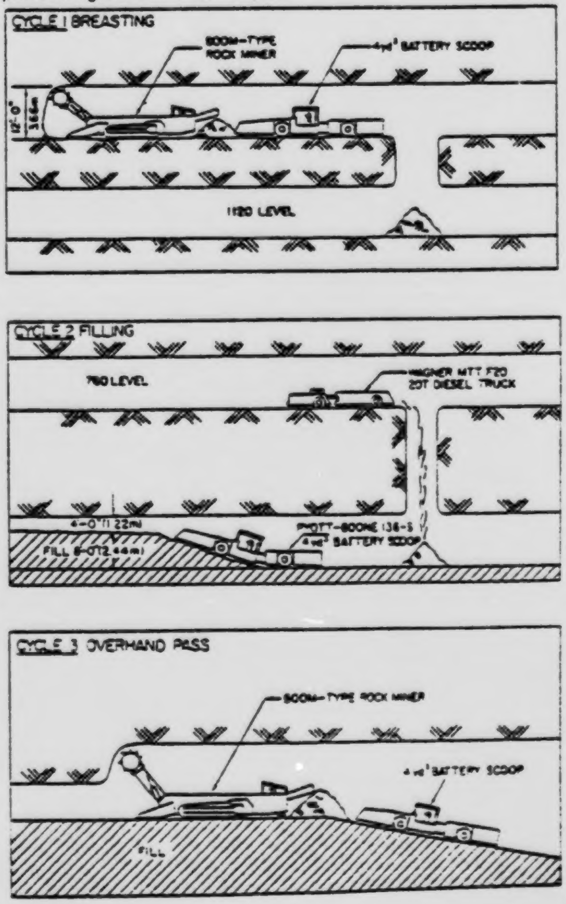
Both machines were brought in on a trial basis to examine their ability to cut high compressive strength rock. Both cut up to 16,000 psi unconfined compressive strength rock successfully. Initially, there were considerable problems with bit design and cutter head lacing. Picks or bits of several different manufacturers were tested for their productivity and ability to withstand high-compressive strength rock. Both conical point attack and flat, kerfing type cutters were investigated.

In general, on the higher horsepowered Dosco, the kerf type cutters gave the best productivity within reasonable bit cost constraints in rocks of under 5000 psi unconfined compressive strength. In stronger rocks, up to 20,000 psi unconfined compressive strength (UCCS), long conical shank point attack bits were necessary to give reasonable production and bit cost.

With the smaller AEC machine, it was found that for rock strengths of up to 7000 psi unconfined compressive strength, small modified plumb bob type point attack bits gave the best productivity and cost. However, at rock strengths rising up to the range of 10,000 to 20,000 psi unconfined compressive strength, heavier modified plumb bob bits were in order (fig. 3). The optimum spacing and angle of attack for all the point attack bits seems to fall in approximately the same range for all of our applications—that is 1¾ in. for bit spacing, with a 44° angle of attack.

The AEC continuous rock miner came equipped with field-interchangeable ripper and milling (auger) cutter

Fig. 5. The sequence of mining involves three cycles: a breasting pass, filling and an overhand pass



heads (fig. 4). Tests with identical cutter motors showed that the ripper had a 25 to 30 percent higher rate of production than the milling (auger) head. For both types of cutter heads, bit consumption, particle size of the cuttings, and amounts of fines and airborne dust were about the same.

In addition to examining each of the machine's ability to cut high compressive strength rock at reasonable production rates, we were also vitally interested in the ability of the machines to selectively mine interbedded ore and waste. Both machines were found to be easily capable of selectively mining beds down to 2 ft in thickness.

In the final analysis of the two apparently competitive machines, ABC developed no preference for one over the other but instead found that the machines were not, in fact, directly competitive in any respect; that is, in terms of weight, productivity, horsepower, price or degree of complexity. It was decided to retain the AEC machines for development work where highly mobile machines with the ability to cut high compressive strength rock in a selective manner was desirable, and it was decided to retain the Dosco TB 600's in the production stopes where maximum productivity while selectively mining high compressive strength rock was desirable.

Ore body serviced from two levels

Mine development work included sinking a 16-ft ID concrete-lined shaft approximately 2000 ft to the east of the eastern extent of the ore body. This was necessitated by the fact that there was no more surface disturbance allowed within Death Valley National Monument for the purposes of mineral extraction. The ore body was located near enough to the eastern monument boundary that the shaft and surface facilities could be placed just outside it, but reasonable access could still be obtained by drifting from the shaft over to the ore body. Drifts were driven above and below the ore body at the 760 and 1120-ft level, respectively.

For each stope, one 7-ft ID man and materials raise, two 4-ft ID ore passes and one 4-ft ID fill raise from 760 level to the stoping level will be pulled. The actual stope layout is in the shape of a skewed H wherein two long parallel rooms are interconnected by a crosscut which alternates position with every other lift. These rooms are approximately 25 ft wide by variable length, approximately 500 ft depending on the strike length of the ore body at that particular stope. The width of the permanent pillar between stopes is 30 ft and the width of the temporary pillar between rooms and the stope is also 30 ft.

Raises placed to aid pillar extraction

The raises are placed so that the temporary pillar between the two rooms of each stope can be extracted via

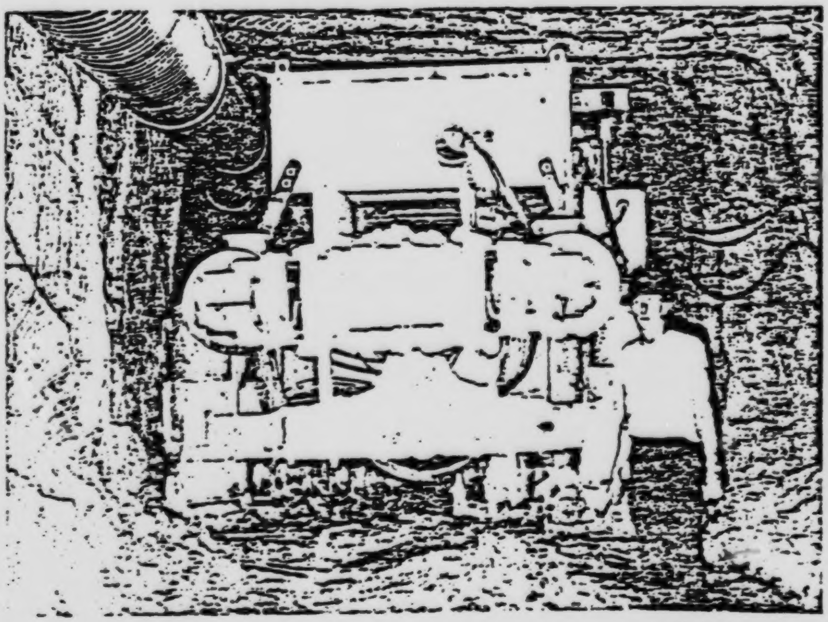


Fig. 6. The American-made Super Roc-Miner (above) is being used in development work where high mobility and an ability to selectively cut high compressive rock are desired. The British-made Dosco TB 600 twin-boom miner (right) has been retained for use in production stopes. The machine is shown here in a development drift

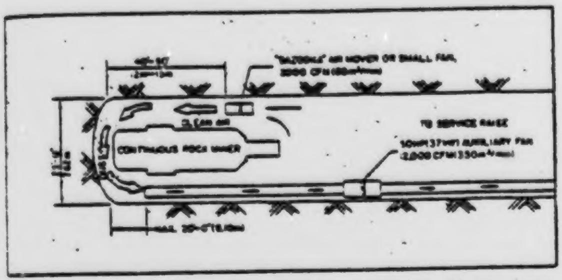


Fig. 7. Face ventilation system. Effective ventilation requires keeping the exhaust intake within 20 ft of the face at all times, pulling a minimum of 12,000 cfm

the same raises. The conservatism of this extraction plan is due to the "zero subsidence potential" requirement imposed by the National Park Service. The H-shaped configuration with alternating crosscuts gives the advantage of four headings per machine for continuous miner place changes, which is important for keeping the machines at the face and producing as much of the time as possible. In addition, the alternating crosscut concept prevents the development of a 300 ft high slice of fill which would effectively cut off future access to the ore remaining in the temporary pillar on the extreme end of the strike away from the access raises.

The sequence of mining is as follows (fig. 5): In the initial breasting pass, the continuous miner cuts the aerial extent of the stope to maximum machine reach, a height of approximately 12 ft. Low profile battery-powered scoops transfer muck from the tail of the continuous miner to the ore passes. If the continuous miner is cutting waste, the scoop deposits the waste in another area of the stope to serve as fill for future lifts.

Down on the haulage level, 30 ft below, a 5-cu yd load haul dump with an ejector bucket picks up the ore or excess waste at the bottom of the ore pass and loads it into 20-ton diesel trucks for transport to the main shaft area. The various types and grades of ore are stored in horizontal storage drifts near the shaft to await hoisting. Once the entire aerial extent of each stope is cut to the maximum height that the continuous miner can reach, the filling cycle begins.

Waste fill is crushed on the surface to minus 2 in. and dumped down a bore hole to a receiver hopper on the 760 level. A 20-ton truck then transports the fill from the receiver hopper to the top of the fill raise above the stope to be filled. The fill is dumped down a raise where it is received by a low profile battery-powered scoop which places the waste in mined out areas in 2 ft thick lifts; four lifts are spread, watered and compacted before cutting resumes. After the placement of a total of 8 ft of fill, the scoop grades the surface of the fill smooth for the continuous miner. The calcareous shale fill assumes a remarkably high strength of up to 750 psi uccs tested in an ordinary 6 x 12-in. test cylinder when it is moistened to approximately 7 percent moisture and compacted to 85 percent of maximum compaction.

Once the filling is completed, the continuous miner climbs the short ramp to the top of the 8-ft thick fill, cutting its way into the back as it goes. The entire aerial extent of the stope is then overhanded to maximum profile and the cycle begins once again.

Good face ventilation critical to successful operation

Since good face ventilation is critical to the successful operation of a continuous miner face, a brief description of the Billie mine ventilation system will be of interest to the reader. The main mine shaft serves as a downcast air way to the 1120 level. Fresh air is coarsed along 1120 level to the bottom of the stope raises. From there, it proceeds up the raises to the 760 level where it returns along the 760 level to two vent raises driven by a 400-hp exhaust fan and 250-hp fan, respectively. Actual face ventilation in the stopes is accomplished through the use of 50-hp auxiliary vent fans and flexible ducting.

It has been our experience that for effective face ventilation of a roadheader type continuous miner, as described here, one needs to pull a minimum of 12,000 cfm from a profile nominally 12 ft high and 20 ft wide. The intake for this exhaust air system must be kept within 20 ft of the face at all times for the system to be effective. This consideration is for fresh air for the men at the face and for dust evacuation. It does not consider gas dilution as the Billie mine is not gassy.

In addition to the exhaust system at the face, we have found it desirable to have a small auxiliary fan located approximately 40 ft behind the face, blowing recirculated air across the face to create turbulence to prevent dust from hanging in dead air space. The volume of the "agitator fan" should not exceed 25 percent of that of the main exhaust fan for best results.

We at ABC believe that this unique blend of methodology best fits our controlling criteria for this deposit. Although numerous problems involved with the blending of equipment and methodology have been encountered, at the time of writing most have been overcome. The first and certainly one of the most critical was developing the hard rock bit technology. This has been successfully overcome in that we now have bit designs and head lacings that give us desired productivity and a bit cost of less than 5 cents per ton.

As is always the case when putting new machinery to work or standard machinery into new applications, there is considerable debugging to do mechanically and electrically. ABC has worked closely with both the manufacturers of the AEC Super Rock and also the Dosco TB 600 to overcome these problems so that the machines are performing with a satisfactory degree of reliability and productivity.

The most perplexing problem of all, which probably is never completely resolved, is that of acquainting the operators with the machinery and with the mining system in general to accomplish the most efficient and effective overall operation in terms of safety and productivity. In general, we at American Borate are satisfied with the result of this rather unorthodox mining system.

Gregory B. Sparks recently joined American Mine Services as senior project engineer from American Borate Co., where he was manager-engineering and planning. Sparks joined American Borate's predecessor, Tenneco Oil Co., in 1974 as mine engineer at its Death Valley, Calif. borate operations. During his employment with the two companies, he also served as mine superintendent-open pit and manager mining division-surface and underground. Earlier, Sparks was a staff mining engineer for Dravo Inc.

Robbins digs on, in Europe and Australia

Two applications are described of work by tunnelling machines supplied by the Robbins company, USA: the 136-204 hard rock rotary machine to be used on the Häusling penstock for the Zillergründl hydro project in Austria; and the tunnel boring machine known as the "Mole", which is steered using laser beam control, working at the Thomson reservoir scheme in Australia.

THE HAUSLING PENSTOCK is an integral part of the feed water system of the Ziller powerplant, which represents a second upper stage to the already existing Mayrhofen station, which is the lower stage of the Zemm powerplants.

The first phase of the Ziller project consisted of consolidating the flow from the Ziller and Boden creeks in the Stillup reservoir of the Zemm scheme, and the addition of a sixth machine set in the Mayrhofen plant. This phase is completed and went into operation June 1, 1977.

Plans for phase two include the creation of the Zillergründl reservoir, the channelling of four creeks, the construction of one 7.4 km-long and one 1.3 km-long penstock and of the Häusling powerhouse. On completion of phase two, connection with the previously built transfer tunnel will be completed. One of the first construction tasks will be driving the Häusling penstock.

The construction contract, worth approximately 50 million Austrian schillings (about $4 million at the current rate) was let after public bidding on October 18, 1978 to the Joint Venture Penstock Häusling, consisting of the companies Ed. Züblin AG, subsidiary Vienna, and Hamberger Baugesellschaft mbH, Linz. The anticipated construction period amounted to 17 months.

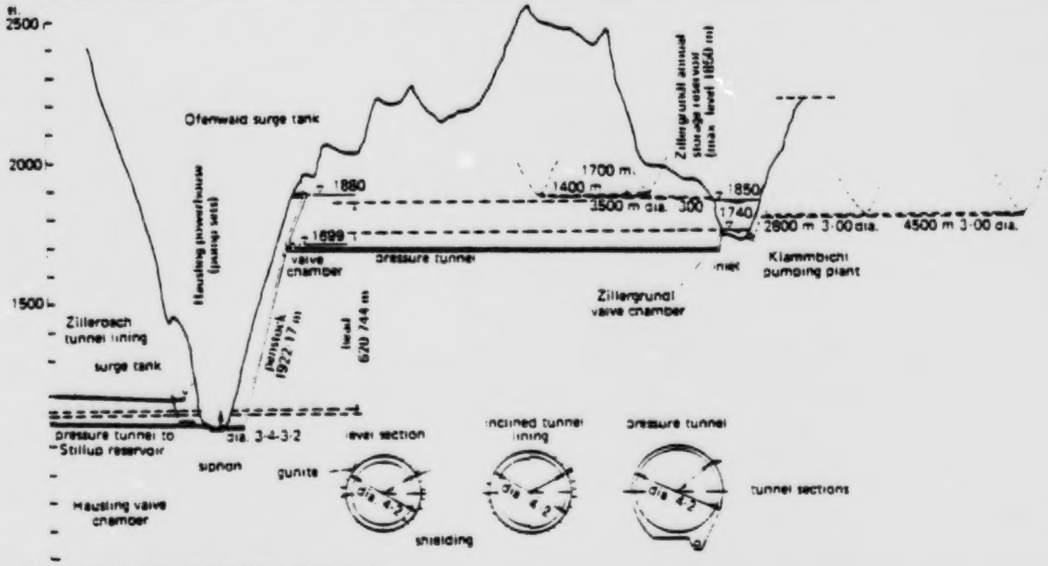
General description of the penstock. The Häusling penstock consists of one 955 m-long section inclined at approximately 42°, and a 400 m lower section with an incline of 1.05 per cent. The distribution pipeline of the powerhouse Häusling branches off the latter. The 42° inclined penstock is extended at an inclination of 33° for 318 m and represents the surge riser. From its upper end, a 254 m tunnel leads into the open. The flat section and the two inclined sections have a circular excavation cross-section of 4.20 m diameter. The line has a curvature radius of 150 m.

Geological conditions. The level section and the lower two thirds of the penstock intersect the so called gneiss of the Schönach basin consisting chiefly of quartz, feldspar and muscovite. The gneiss is expected on the basis of experience gained in building the transfer tunnel on the opposite slope as massive, homogeneous and, in most cases, with few cracks.

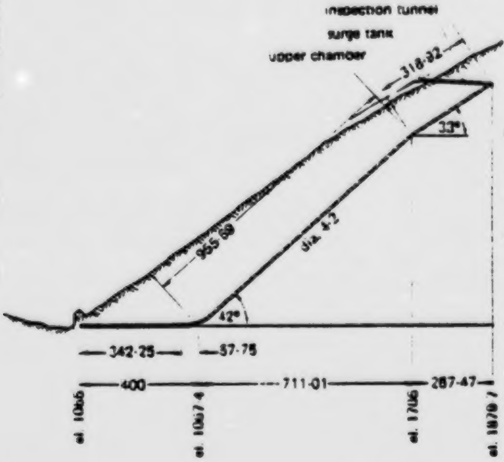
The upper third of the penstock intersects the porphyric gneiss belonging to the central gneiss. The contractors do not expect any difficulties as a result of the geological formation.

A 2-6 m slate band could be observed in the boundary zone between the rock zones described, at an elevation of approximately 1630 m—possibly representing the boundary layer between the two gneiss areas spreading out at the surface, so that one can expect brittle to strongly brittle rock over a length of several tens of metres.

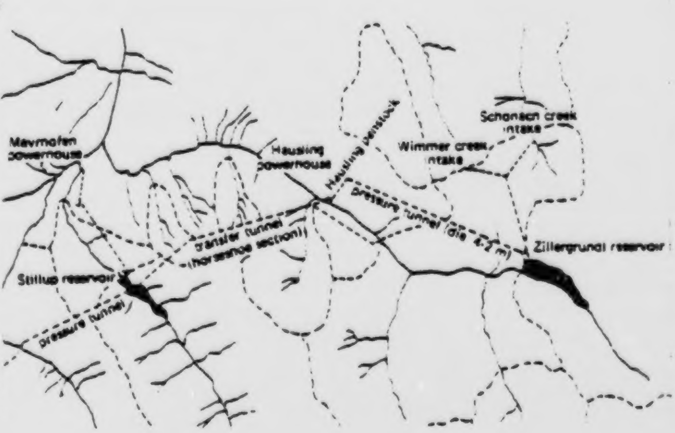
In general, on the basis of geological knowledge, the penstock excavation, with the exception of individual brittle sections, can be considered as solid.


m.
2500
2000
1500
Ofenwald surge tank
Häusling powerhouse (pelton sets)
1880
1699
valve chamber
pressure tunnel
penstock 1922 17 m
head 620 744 m
Zillerbach tunnel lining
surge tank
pressure tunnel to Stillup reservoir
dia. 3-4-3-2
siphon
Häusling valve chamber
Zillergründl annual storage reservoir (max level 1850 m)
1700 m.
1400 m
3500 m dia.
300
1850
1740
2800 m 3·00 dia.
4500 m 3·00 dia.
inlet
Klammbichl pumping plant
Zillergrundl valve chamber
level section
gunite
dia 4·2
shielding
inclined tunnel lining
dia 4·2
pressure tunnel
dia 4·2
tunnel sections

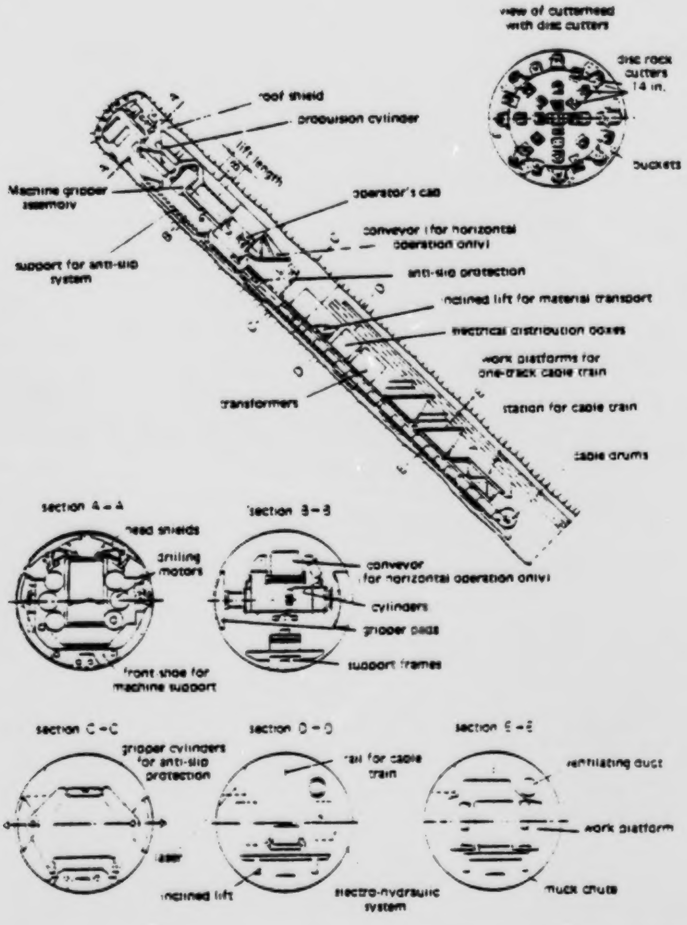
Longitudinal section through the Ziller powerplant.


inspection tunnel
surge tank
upper chamber
318·32
33°
955·59
dia. 4·2
42°
342·25
57·75
400
711·01
287·47
el. 1066
el. 1067·4
el. 1706
el. 1878·7

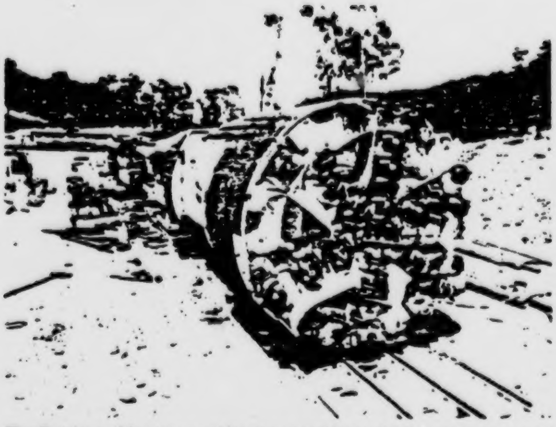
Longitudinal section of the Häusling penstock.


Mayrhofen powerhouse
Häusling powerhouse
Häusling penstock
Wimmer creek intake
Schönach creek intake
pressure tunnel (dia 4·2 m)
Zillergrund reservoir
transfer tunnel (horseshoe section)
Stillup reservoir
pressure tunnel

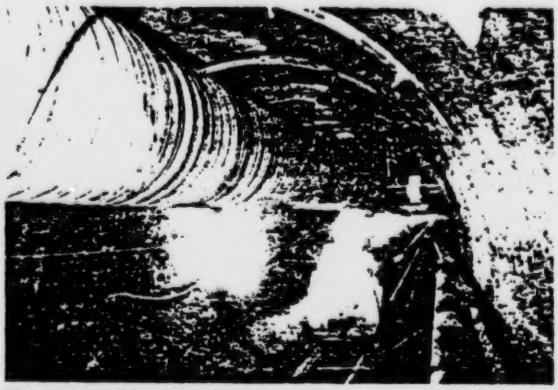
Site plan of the Ziller powerplant.


view of cutterhead with disc cutters
disc rock cutters 14 in.
buckets
roof shield
propulsion cylinder
lift length
Machine gripper assembly
operator's cab
conveyor (for horizontal operation only)
support for anti-slip system
anti-slip protection
inclined lift for material transport
electrical distribution boxes
work platforms for one-track cable train
transformers
station for cable train
cable drums
section A – A
head shields
drilling motors
front shoe for machine support
section B – B
conveyor (for horizontal operation only)
cylinders
gripper pads
support frames
section C – C
gripper cylinders for anti-slip protection
laser
inclined lift
section D – D
rail for cable train
electro-hydraulic system
section E – E
ventilating duct
work platform
muck chute

Longitudinal section through the Robbins tunnel 136-204.



The Robbins 'Mole' tunnel boring machine, which was transported from the Port of Melbourne in sections by low loaders in various stages of assembly.



The thin line of the laser beam can be seen here, which controls the machine's steering alignment. The rail tracks are extended as excavation proceeds, linking the working face with the tunnel portal.

Construction method

From the starting point, a starting tube approximately 30 m long is established by the conventional excavation method. Succeeding this, the mechanical drive with a tunneller can proceed. A Robbins 136-204 type machine will be used, which will probably be assembled in August at the site. Then the drive can start. The necessary shotcrete protection must not project more than 5 cm into the theoretical profile. In cementing possible subsequent overbreak, an excess radius of 5 cm will be acceptable. After the tunnel is driven, it will probably be given a concrete lining in situ with foil sealing and in areas of higher inner pressure a shielding in the form of steel sheet lining. The muck accumulated in driving the steep section is washed out with water. In the lower bending point section, during transition into the level section, the muck is separated from the water and then carried with muck cars to the outside.

Before return to the creek bed, the water is treated appropriately.

The Mole in Melbourne

The "Mole", a tunnel boring machine also supplied by Robbins, USA, arrived in Melbourne, Australia in January 1979 for work on the 6 km Thomson-Yarra tunnel, part of stage 3 of the Thomson reservoir dam project. The machine is expected to finish the job by May 1980.

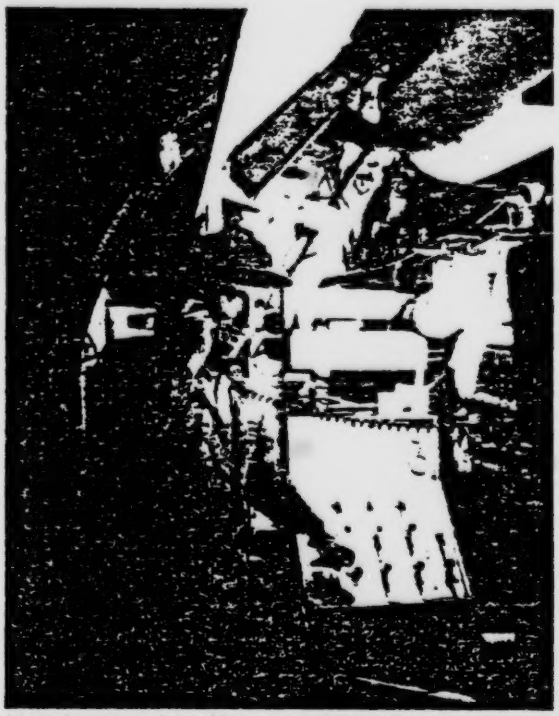
A laser beam controls the Mole's steering alignment. A light-sensitive target is used, so that a precise print-out of the machine's position can be obtained whenever necessary. Unless the ground is hard on one side and soft on the other, laser control enables the contractors to steer to the nearest 2 cm.

Six 93 kW electric motors drive the 4.1 m-diameter 5.2 rev/min cutting head as it bores through 4 m of rock per hour. That rate of cutting produces up to $120\ m^3$ of broken rock. Six scoops in the rotating head pick this up and deposit it on a conveyor belt, which drops it into $8\ m^3$ rail cars. These take it to the surface for disposal.

The Mole and its train of trucks are 100 m long, weigh 120 tonne, and have their own power and water supply.

The cutting face of the "Mole" has 29 steel discs, worth about $2000 each. These rotate as the machine head revolves, cutting the rock like a glass cutter. The Mole is stabilised by steel grippers, held against the sides of the tunnel by hydraulic rams. Oil pumped under pressure into cylinders causes propelling rams at the back of the cutting head to drive the cutters deep into the rockface with a thrust of 800 tonne, so the machine head can progress.

The operator's cabin is set back 11 m from the cutting face. From this position, the operator can monitor almost every aspect of the tunnelling action. He steers the machine, checks on the performance of each motor and hydraulic ram, and has a good overall view.



All the functions of the 'Mole' are controlled from the operator's cabin, shown here; this acts as a nerve centre for the machine, from which phone links are maintained continually with the workers outside.

The cutters on the Mole are mounted on a head which was designed and patented by Melbourne and Metropolitan Board of Works, the patent was later sold to the American company. An engineer employed by Robbins in Seattle designed the cutters.

The Board of Works already has experience of tunnelling schemes in Australia. It successfully tendered for a $2.8 million contract to bore two 125 m tunnels for Melbourne's underground railway. In addition, the Board's experience in tunnelling the South-Eastern and Dandenong Valley trunk sewers meant that many of the likely technical problems had been resolved before it tackled the Thomson project.

Another major plant item on the Thomson project is the Robbins raise boring machine. Bought second-hand in 1978 for $175 000, this vertical borer is used to cut a 1.2 m vertical hole through 100 m of rock. This is part of the intake works, and will permit selective withdrawal of water from the reservoir.

Description of Thomson project

The average depth of the Thomson storage will be 75 m, and the water temperature can vary naturally between 8-15° Celsius from top to bottom.

Water released downstream for irrigation which is too cold (or too warm) would adversely affect the river ecology, in particular the breeding behaviour of fish. It is therefore important that water be drawn from a level at which its temperature is compatible with downstream temperatures.

To ensure the best quality water from the Thomson to the Upper Yarra reservoir and the Melbourne supply, water will be taken from below 5 m to avoid any algae or other growths. The Thomson river project will also benefit the people living downstream of the dam. It will enable releases from the reservoir at times of low river flow, and regulation of the river when heavy rains threaten to cause floods.

Work on the Thomson-Yarra tunnel and the main intake by the two Robbins machines is one important part of the overall project, which is scheduled for completion in 1984. The Thomson dam itself requires 16×10^6 m^3 of earth and rockfill, forming a wall to hold back the water of this reservoir which will extend for 23 km upstream.

The project has been carried out in three stages.

The first, completed in 1974, was the diversion of water from the Upper Thomson river into the Yarra system through a 20 km tunnel. The second, a diversion weir on the Thomson river and an 11 km extension of the original tunnel, was commissioned by the Premier, Mr Hamer, in November 1978.

The third and final stage is building a major earth and rockfill dam 165 m high on the Thomson river and extending the Thomson-Yarra tunnel another 6 km. This stage is due for completion in 1984.

The Thomson reservoir catchment will be 48 700 ha and maximum usable capacity 1.1×10^6 Ml.

Comparable in size to the giant Talbingo dam in the Snowy Mountains Scheme, the Thomson reservoir project began with site investigations in 1974. Work is progressing well, both above and below ground. Board staff are working under what are often difficult conditions to make Melbourne virtually drought-proof by the late 1980s, when the reservoir is expected to be full for the first time. □

This article is based on a publication by Joint Venture Penstock Häusling, comprising Ed. Züblin AG, Civil Engineering Contractors, Vienna, and Hamberger Baugesellschaft, Linz.

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Operation of Wirth Tunnel Boring Units in Granite

by

W. HILDEBRAND*

SUMMARY

The case histories of boring three tunnels with horizontal and inclined sections in hard, granitic rocks with compressive strengths up to 2 800 bar, using Wirth tunnelling machines are presented. The relative performances of insert, gear and disc-type cutters are discussed. Cutter costs proved to be of the order of R15/m³ and rates of advance in excess of a metre per hour were obtained. Based on these experiences recommendations for future equipment are made and the strategy for the construction of two road tunnels near Lucerne by boring, followed by two stages of reaming, is discussed.

INTRODUCTION

Tunnel boring has attracted worldwide interest as an increasing number of incline tunnels, road tunnels and shafts are being driven without explosives. For the most part, these tunnels have been bored in formations having moderate to low compressive strengths, but rotary tunnelling equipment is being developed to the stage where all rock formations can be bored at costs competitive with those of tunnelling by blasting.

The extraordinary development of rotary tunnel-boring units in recent years is due not only to the wishes or the insistence of engineers for new techniques or machines, but also to the continuously increasing numbers of all types of tunnels.

To date, most tunnels have been bored in comparatively soft rock. Tunnel-boring units for soft to medium-hard rock have been discussed in technical journals and will not be described in this paper. A breakthrough in driving tunnels, galleries or shafts by boring will have been achieved when it is possible to bore tunnels in the hardest rock formations, such as granite, with reasonable cutter costs.

In recent years several tunnel-boring units have been used on projects in granite and, from the results obtained, it is evident that this breakthrough has been reached from the tunnelling, engineering and economic points of view.

This claim is substantiated by the records of tunnelling projects in granite by machine boring described below.

CASE HISTORIES

Zemm hydro-electric power station

In 1967 the Wirth tunnelling unit TB I-214 was used to bore a tunnel in very hard granite as part of the Zemm hydro-electric power station project in Austria.

The diameter and length of the gallery were 2.14 m and 263 m, respectively, and the formation was a gneiss with a compressive strength of approximately 2 600 to 2 800 bar.

At the time of construction many specialists viewed the attempt to bore this tunnel sceptically. Despite a number of mistakes, much experience was gained which was of value in later projects.

Grand Emosson hydro-electric power station—Corbes tunnel

The experience obtained at the Zemm project formed the basis for the use of a Wirth tunnel-boring unit, type TB II-300, to drive the Corbes pressure tunnel during the construction of the hydro-electric power station, Grand Emosson, Switzerland. The tunnel consists of a horizontal section of 350 m followed by an inclined section of 1 140 m with a gradient of 65 per cent (Figure 1).

The formation in the inclined section was uniform Vallorcine granite, with a high quartz content and a compressive strength of 2 400 bar.

The tender documents specified that the tunnel had to be driven conventionally, but the following favourable conditions for tunnel boring existed:

1. There were no extreme inclinations.

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- ii) A uniform rock formation existed with probably a few fault zones only.
- iii) The civil engineering contractor had a very positive attitude towards tunnelling by boring.

Economic considerations finally decided the Swiss civil engineering company to use a boring unit on the project.

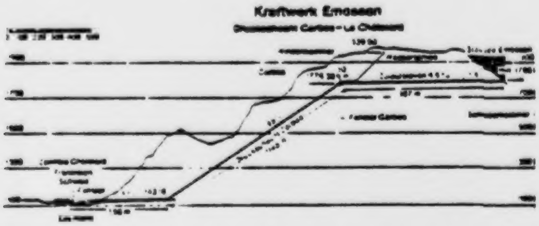

Figure 1 — A section through the Corbes — Le Châtelard hydraulic tunnel for the Grand Emosson power station.

In a horizontal tunnel the investment in the boring unit and cutter costs can be higher than the savings made on wages and material costs. In an inclined tunnel, however, the working conditions are such that savings can be made which become very important in reaching the point where tunnelling by boring becomes economical:

- i) The accurate profile results in lower concrete costs.
- ii) Good control of the profile requires no profiling afterwards.
- iii) Better support to the casing or lining is provided against the rock.
- iv) Less roof-bolting or rock securing is needed.
- v) Transport of cuttings in the tunnel is easy.
- vi) Use of a casing or lining with a smaller wall thickness is possible.
- vii) Installation of the casing or lining is easier, and
- viii) Fewer personnel are required.

The boring equipment was delivered on site and moved into the inclined position in August, 1968. To enable the unit to be anchored it was erected in an excavated and concreted part of the inclined tunnel. A most important part of the machine was the non-returning safety device to prevent it sliding backwards.

PERFORMANCE. Boring started on 1st October, 1968, and the unit was in operation continuously, with only minor breakdowns, until 1st September, 1969. After allowing for down-time over Christmas, 1968, and other operational down-times not connected with the unit, the inclined tunnel was completed three months ahead of schedule. This amounted to an advance of approximately 30 cm per effective boring hour. Boring in normal uniform formation without faults was carried out with 210 tons thrust on the pressure gauge from which 50 tons has to be deducted for the weight of the unit and friction, leaving an effective thrust of 160 tons on the cutters, or 5 tons per cutter. The cutters were designed for a continuous load of 10 to 12 tons, thus giving a safety margin which resulted in improved running times of the cutters.

With a rate of progress of 50 cm hour, one stroke of the machine of 75 cm took about 90 to 95 minutes. Resetting the unit with the assistance of the safety device took about five minutes. An effective boring time of 60 to 70 per cent of the total time could be accomplished, even with delays such as cutter-control every two days and greasing or changing cutters being taken into consideration.

The average performance, calculated after completing the drilling operations, amounted to 5.30 m per working day of two shifts. The maximum advance in one day amounted to 15.75 m and the maximum advance in a month amounted to 140 m.

OPERATING CREW. On site, the following personnel were required per shaft:

One operator.
Three line-workers for back-up work.
One winch man.
One engine driver.

This crew of six men took care of all the boring operations. The back-up group of three men had a very important job to do, as they were responsible for extending the power and supply lines: the capacity of the boring unit could be maintained only if the back-up group kept pace with the unit.

CUTTERS. The cutter costs are the most important factor in drilling hard rock; more so in drilling extremely hard granite with a high quartz content, and success or failure is completely dependent on these costs.

On this project a total of 114 cutters were used on a total length of 1 140 m. The last set of cutters could, however, be used again and a number of the discarded cutters could be repaired. Including spares and spare bearings, the total costs were R15 m^3.

However, as the result of tests conducted during these operations, cutters can now be supplied which can be used for longer periods. A considerable reduction in manufacturing costs of cutters has also been achieved. Cutter costs have been reduced to such an extent that savings of up to 20 per cent in costs; have been achieved and improvements in bearing design can decrease the costs further. The cutters will always remain the most important factor in boring hard rock, and careful maintenance and a tight greasing schedule are most important.

It is imperative that a record of numbered cutters and locations on the boring-head be kept when cutters are replaced: especially when new and used cutters are mounted on the head. After a few millimetres wear on the cutter inserts the boring efficiency is reduced, but the cutters as such need not be discarded as useless. The efficiency of the insert type of cutters can be improved greatly if they are removed and repaired in time, or exchanged for new inserts. A certain amount of experience is required to judge cutter life and this can be gained only from further operation.

At first the cutter cost of R15 m^3 seems to be quite high, but it should be borne in mind that 2.45 m^3 of concrete is saved per metre of tunnel length and a major part of the cutter costs is offset in savings on explosives, detonators and drill steels used in conventional tunelling.

The total time required to drill the 'Corbes tunnel' was approximately nine months and the breakthrough was made on 1st September, 1969.

Notwithstanding many doubts initially, the project was an unparralleled success and all parties concerned were very proud and satisfied with the performance.

Grand Emosson hydro-electric power station — Barbarine Tunnel

The 'Barbarine Tunnel' was bored as part of the Grand Emosson project in which water from the Emosson reservoir at 1 770 metres above sea level is fed through a pressure tunnel to the hydro-electric power station at Châtelard-Giétroz at 1 225 metres above sea level (Figure 2).

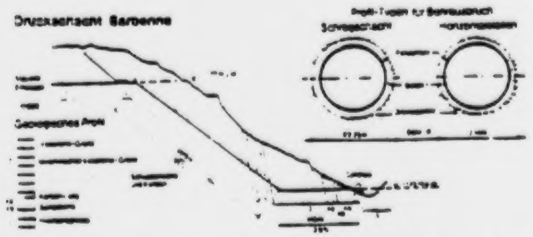

Figure 2 — Sections through the Barbarine pressure tunnel of the Grand Emosson project.

The horizontal section of this tunnel is approximately 400 metres long with an inclination of 0.5 per cent and a diameter of 2.14 metres. The inclined part of the pressure tunnel has a length of 1 020 metres, with an inclination of 80 per cent and a diameter of 2.25 metres. The formation to be bored consisted of mylonitized Vallorcine granite in the horizontal section and a homogeneous Vallorcine granite with a compressive section of 2 400 bar in the inclined part.

Boring operations were carried out by means of the Wirth tunnel unit type TB 1-240, the horizontal section being started on 8th October, 1968.

After the horizontal section was completed the unit had to be adapted for inclined operation and then moved into the inclined position.

Following favourable tests in the horizontal section it was decided to bore the inclined section with disc-cutters from 630 metres. Up till then only insert type cutters had been regarded as economical for drilling the Vallorcine granite, but the use of disc-cutters appears to be very promising for the reduction of cutter costs (Figure 3). The cuttings changed rapidly in size as soon as the disc-cutters were used.

The boring rate which had been 0.65 to 0.95 metres per hour was increased to 0.7 to 1.1 metre per hour. The total thrust of 105 to 125 tons did not increase the power requirements of the unit and the wear of the disc-cutters remained within acceptable limits.

With the disc-cutters, lives of up to 150 metres were obtained and it should be borne in mind that the discs are relatively cheap and that the bearings could be used again.

A further considerable reduction in cutter costs will be achieved as soon as all granite can be bored by means of disc-cutters.

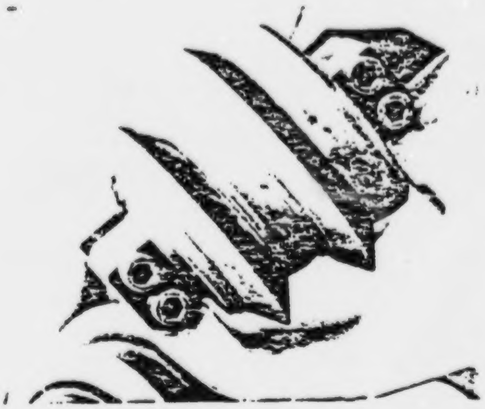

Figure 3 — A disc-cutter for use in hard rocks.

FUTURE EQUIPMENT BASED ON PAST EXPERIENCE

From the wealth of experience and results obtained during actual tunnel boring operations in granite, many conclusions have been drawn.

For drilling in hard to very hard rock, and not only granite, certain basic requirements are necessary and boring tools or cutters must be chosen accordingly.

The supports for these cutters should be arranged on the boring-head in such a way that the cutters can be mounted or dismantled rapidly.

Cutter supports should be made so that, without any modification to the boring-head, the following cutters can be used:

Insert type cutters;
Disc-cutters;
Tooth cutters;

so that when the rock formations alter they can be changed rapidly (Figure 4).

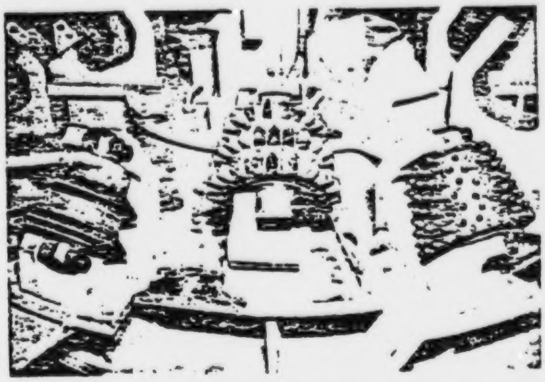

Figure 4 — A Wirth boring-head showing disc, gear and insert cutters.

Experience has shown that at least the insert type of cutters have a much longer life in granite than expected. The target, however, is to increase bearing life to at least 1 000 running hours, to equal or even to exceed the expected life of the actual cutter which is nearly 500 metres.

ROAD TUNNELS — LUCERNE

In Switzerland further extensions are to be made to highway N-2 in the Lucerne area and two adjacent tunnels each with a diameter of 10.46 metres have to be driven.

The order for the design and construction of the boring unit for this tunnel was given to Alfred Wirth & Co.

A pilot tunnel is to be bored using a unit with a diameter of 3.5 metres. This will afterwards be reamed to a diameter of 7.70 metres, followed by a second unit to ream the tunnel to the required diameter of 10.46 metres.

By means of this system, all the required information about the rock to be bored will be obtained and the contractor will be able to cope with fault zones, water influx and other possible problems before reaming commences.

The diameters of the heads of the tunnel boring and reaming units have a certain range, and with this combination the contractors will have the ability to tender on drilling projects with a very wide range in diameters and also to use one of the three units on a single tunnelling project only.

Wirth boring-heads are all designed in such a way that rapid changing of cutters is possible to meet changing geological conditions.

The system with reaming units can be used in tunnelling horizontally or on the incline, thus increasing further the capabilities and adaptabilities of the complete unit.

CONCLUSION

Development of rotary boring tunnelling equipment has seen an extraordinary progress in recent years and the examples summarized in this paper demonstrate that boring operations in all types of rock formation have a great future.

EQUIPMENT AND MAINTENANCE

Mine applications of tunnel boring machines

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ABSTRACT

Tunnel boring machines have been tried in underground mining applications as early as 1957. Although tunneling machines hardly threaten to supplant traditional drill-blast-muck operations for drifting, the latest modern machines offer substantial improvements in progress rates over traditional methods under certain conditions.

An evaluation guide is offered which identifies conditions that favour machine boring. Different types of costs related to the use of a tunnel machine are discussed. The advantages of the tunnel machine over the traditional method are reviewed.

Introduction

The first tunnel boring machine ever used in a mine was in Quebec in the year 1957. Although less than successful at the time, this experiment served to illustrate the mining industry's willingness to try new ideas. Since 1957, the manufacture of tunnel boring machines has developed into a full-size industry. Most contractors in Europe and North America will now consider the use of a tunnel boring machine first on any long tunnel that can have a circular section.

The modern tunneling machine offers a great potential to the mining industry. Just as the blasthole and raise drill machines have revolutionized mining in the last decade, it is expected that tunnel boring machines will make an equal impact in the future.

This is not to suggest that tunnel boring machines will replace explosives in the foreseeable future, but they can supplement drill-and-blast operations. In areas where long drifts, inclines, declines or cross-cuts are needed in a hurry, the tunnel boring machines offer the fastest way to a production return on investment. Figure 1 shows the range of production that two types of tunnel machines have produced as compared with conventional excavation methods in various ground conditions.

Rotary hard-rock tunnel boring machines (Fig. 2) commonly produce progress rates in excess of 20 metres per day in uniform ground from 300 to over 2,000 kg/cm² in unconfined compression strength. These averages are for the completed tunnels. One 7.16-metre-diameter tunnel boring machine just completed a 10.7-km tunnel in hard rock of 700 to 2,800 kg/cm². The last eight months of production averaged 30 metres per day.

Two principal elements make up such progress (Fig. 3). The first is the advance rate of the machine and the second is the utilization of the machine by the operator. An advance of 2 metres per hour will give an average progress of 8 metres per eight-hour shift. Historically, a given tunnel machine will be 80 to 85 per cent available for operation, but is only utilized about 50 to 60 per cent of the available shift time. A 10 per cent increase in utilization, under normal conditions, would total 9.6 metres per shift. However, a 10 per cent increase in penetration rate would only total 8.8 metres per shift.

ROCK STRENGTH	FLUID TO 300 kg/cm²	300 kg TO 900 kg/cm²	OVER 900 kg/cm²
SUPPORT REQUIRED	FULL	MINIMUM	NO
SHIELD EXCAVATOR	2 TO 5	20 +	3 TO 5
ROTARY MACHINE	0	20+	20+
CONVENTIONAL	2	6 TO 8	12

FIGURE 1. Average progress rates (metres per day) for different ground conditions.

This exercise illustrates the importance of well-trained and competent underground crews. Figure 4 shows the relative importance of equipment, crew and geology for tunnel machines and conventional excavation methods. One can control the equipment and crew selection, and the use of a tunnel machine can reduce geologic hazards considerably. To increase utilization, one must consider the tunneling system as a whole. Some shifts will achieve 100 per cent machine system utilization. The machine backup and muck handling must be sized to handle such sustained progress. If the above machine was 4.5 metres in diameter and the rock swell factor 1.8, nearly 230 cubic metres of waste rock would have to be removed from the heading and transported to the surface in one eight-hour shift.

Economic Considerations

Project Considerations

Full-face rotary machines should be considered only if they can provide an economic or time advantage over conventional mining methods. Generally, this must be calculated for each individual project. Most machine manufacturers and many contractors can provide economic and time studies to evaluate the practicality of a tunnel machine on a given project.

"Rules of thumb" can often be quite misleading. However, the following are offered to provide a list of items to be evaluated when considering a tunnel machine.

- Tunnel length—greater than 2 km
- Tunnel diameter—2.1 to 6.1 m
- Rock strength—under 2,800 kg/cm²
- Geologic structure—bedded, jointed, blocky, etc.

Keywords: Equipment, Boring machines, Tunnel boring, Underground mining, Machine utilization.

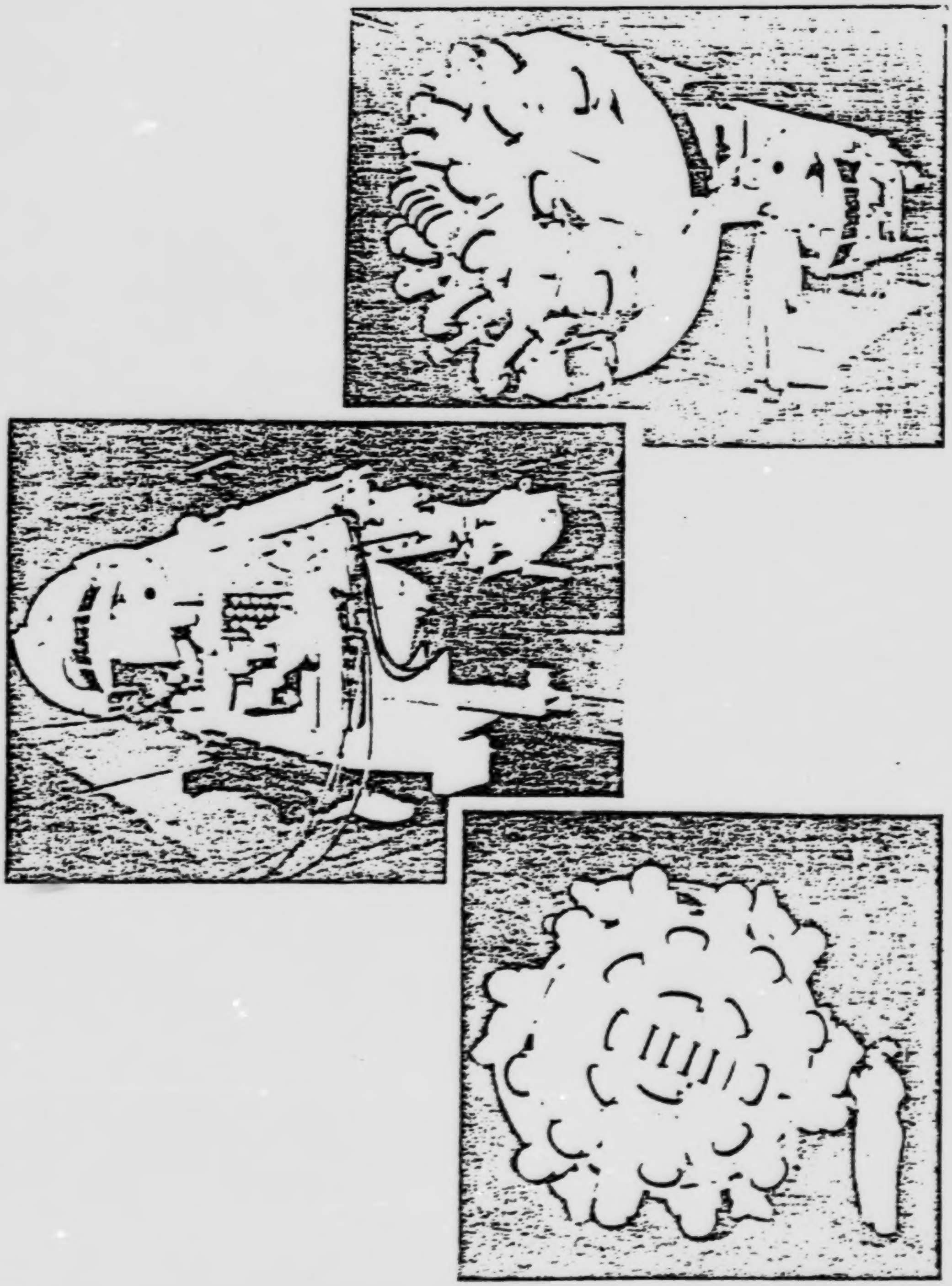

FIGURE 2. The rotary rock borer—open type.

- Tunnel turns—over 100 m radius
- Tunnel grade—18 to +30 per cent
- Maximum water inflows (at face)—1 m^3/minute/m of diameter
- Maximum water pressure—under 15 bars
- Waste or muck disposal rate—Can system support sustained production from the TBM?
- Maximum ground load—two tunnel diameters
- Power (electrics)—Is adequate power available? Can it be provided?
- Shaft size and hoisting capacity—Can it handle the largest components? Can the high muck disposal rates be handled?

There exist many exceptions to the above "rules". As an example: if a contractor has a used machine that fits the specifications for the project, he may well be able to justify a machine on a tunnel shorter than 2 km. Another example is the 100-m turn radius. Special machines have been designed to turn in a radius as short as 30 m. Larger water inflows, heavier ground loads and harder rock can be tolerated for short distances on almost any project, although they may cause delays and higher costs.

Tunneling costs can be broken down into two basic categories—fixed and variable costs.

Fixed costs include such items as equipment amortization, including some spare parts, jobsite installations and tunnel services such as track, fan line, power cable, water and air lines, lighting, etc. Although tunnel supports are a variable cost depending on rock conditions, in a simplified case where constant ground conditions are assumed, they can be considered a fixed cost per foot of tunnel, similar to the tunnel services, as their cost is independent of rate of advance.

The variable costs include labour, power, overhead and cutter costs. Cutter costs might be considered to be fixed for any given rock condition, but prove to be quite variable in practice because they are dependent on the machine penetration rate and maintenance procedures, which are both a function of the performance of tunnel personnel.

Machine-bored tunnel costs are shown graphically in Figure 5 as they relate to the tunnel's over-all average advance rate. It is apparent from this illustration that factors which achieve a major shift in the over-all average advance rate also have a major effect on tunnel costs. The factors which have often had the most serious effect on over-all average advance rate are:

a. Unexpected large variations in tunneling conditions. This might include such things as major fault zones, squeezing plastic clay and large water inflows.

b. Unexpected variations in rock strength abrasiveness and hardness, which will affect penetration rate, downtime for cutter changing and cutter cost.

c. Major breakdowns which can be the result of either manufacturing problems or operating techniques, such as erratic steering, advancing with worn-out cutters and poor maintenance.

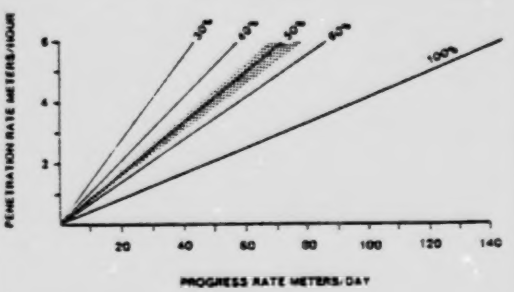

FIGURE 3. Tunnel machine utilization.

	EQUIPMENT	CREW	GEOLOGY	
TUNNEL MACHINE	1/3	1/3	1/3	= 1
CONVENTIONAL	1/4	1/4	1/2	= 1

FIGURE 4. The proportional importance of equipment, crew and geology for both conventional and machine excavation methods.

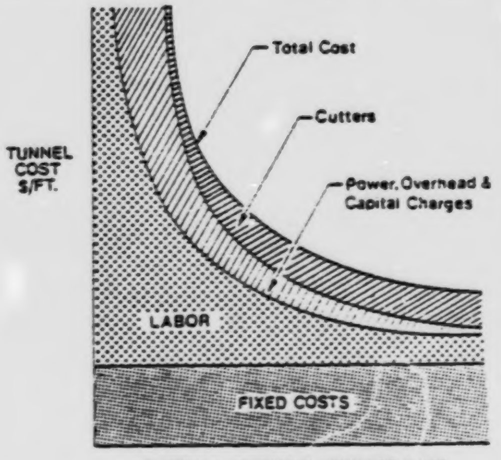

FIGURE 5. Machine-bored tunnel costs vs average advance rate.

Machine and Accessory Cost Considerations

New full-face tunnel boring machines require a very large capital outlay. In 1979, a 4.5-m (14.8-ft) diameter machine cost from $2,300,000 to $2,500,000, depending on the horsepower and ground support systems required. Backup systems, gantry conveyors, passing switch, transformers, air scrubbers, etc., would run to an additional $300,000 to $500,000. Usually it would require several kilometres of tunnel to write off this initial cost.

In most mining operations, 4 to 6 km of tunnel, development headings and/or haulage ways can be enough to justify the capital outlay for a tunnel machine. Where on-going tunneling is contemplated, a reasonable write-off period for a full-face rotary rock tunneling machine would be eight years or 12 km of tunnel. Providing that the machine has been maintained and repaired, it would then have a salvage value of 25 to 50 per cent of the original cost. In addition to these capital costs, spare parts, insurance spares and rock cutter costs must also be considered.

Spare parts costs appear to vary somewhat with the abrasiveness of rock. For estimating with 1979 dollar prices, these costs could be from $0.75 to $1.50 per cubic metre. Insurance spares are those which may not be used routinely, but do require long manufacturing lead times. To prevent long production delays, insurance spares amounting to approximately 10 per cent of the machine value are generally recommended.

As with any heavy machinery, when a good preventative maintenance program is employed, the machine's useful production and life can be greatly extended. This writer knows of at least 33 tunnel machines that have made over 6 km of tunnel each. Of these, more than eight machines have made over 12 km. Many of these machines are working or available to work at the time of this writing.

Rock cutter costs can vary considerably with rock conditions and strength, and the type of rock cutter employed. Figure 6 is for very rough estimating only. Most tunnel machine manufacturers and some contractors offer specialized rock testing services. This rock testing is compared with previous tunnel machine experience in similar rock conditions and an estimate of cutter type and costs and machine advance rates can be made.

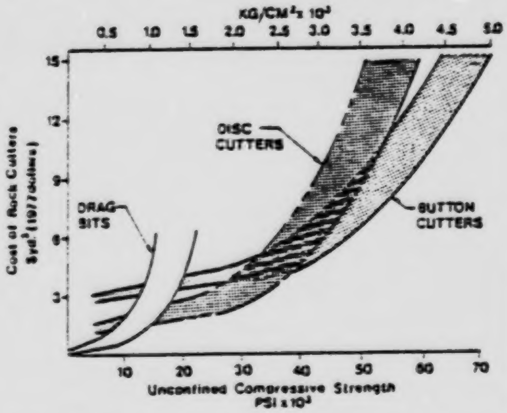

FIGURE 6. Typical range of cutter costs (in rock of average toughness and abrasion).

Few special tools, if any, are required with most tunnel machines. Many of the machines on the market today have been designed to be assembled and serviced with standard hand and power tools.

Tunnel Machine vs Drill-and-Blast Methods for Tunnel Excavation

Full-face rotary tunnel machines offer many advantages in economy, safety and speed over conventional drilling blasting.

When conventional drill-and-blast methods are used to develop a heading, explosives disturb the ground. This will open cracks, which will increase the likelihood of rock falls and expose more ground to water inflows. The nature of fracturing rocks with explosives leaves much overbreak and some "tights", depending on the skill of the underground crew. Loose rock must be scaled down, which at best is a hazardous operation.

Full-face rotary tunnel machines, on the other hand, disturb only the rock to be removed. This leaves a very smooth and circular tunnel wall offering little resistance to the flow of ventilation air. Ground support requirements are reduced, overbreak is minimized and "tights" are eliminated with the tunnel machine. Pillars can be constructed with smooth uniform surfaces and rock stress concentrations are usually reduced with these machines.

In addition, tunnel machines can develop headings at three to ten times the conventional rates, provide much safer and more desirable underground working conditions and require less skilled labour than do drill-and-blast methods.

The background to the National Geological Surveys Committee

The National Geological Surveys Committee was established early in 1979 to formalize communication between the various provincial geological surveys and the Geological Survey of Canada. Informal working arrangements have existed in the past between the GSC and individual provinces, and the provincial geologists themselves have a Standing Subcommittee under the umbrella of the Provincial Mines Ministers Conference.

In response to a recommendation of the Canadian Geoscience Council, a representative of the Geological Survey and members of the Standing Subcommittee met in March 1979 and proposed the formation of an independent committee—the NGSC. The first meeting was held in Quebec City in May 1979 and subsequent meetings have been in Ottawa in October 1979 and Halifax in May 1980.

The NCGS is made up of one member for each of the provinces and the two northern territories which are represented by two regional staff members of the Department of Indian and Northern Affairs, and one member from the Geological Survey of Canada, the chief geologist. In addition, the director general of the GSC serves as chairman.

The Committee's objectives are to facilitate: coordination of the work of geological surveys in Canada; establishment of consistent national standards; evaluation and adoption of new methods and ideas; intergovernment access to professional geoscientific expertise.

The Committee has already established subcommittees and working groups to prepare recommendations on the following matters:

1. Fundamental Research in Mineral Deposits.
2. Exploration Technology.
3. Geoscience Standards, including (a) geological map symbols and abbreviations,(b) identification of geographic locations, (c) exploration geochemistry, (d) exploration geophysics and (e) standards for machine-readable exploration data.
4. GEOSCAN. Responsibility for recommending management policies for this National System for Geological Information was transferred from the Mines Ministers Conference in late 1979.

Other topics are under active considerations, but the Committee would welcome additional suggestions from members of the geoscience community at large. These should be sent to: Director General, Geological Survey of Canada, 601 Booth St., Ottawa, Ontario, K1A OE8.

Application of hard-rock boring machines in deep South African gold mines of Anglo American Corporation

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Synopsis

The layout of a typical deep-level Anglo American Corporation gold mine, the mechanization programme for the gold mines and the hard rock boring programme are described.

Tunnel-boring experience at Free State Geduld gold mine is detailed. Although much has been achieved in introducing a tunnel borer into a gold mine, the Wirth machine has not yet been fully adapted to enable acceptable advance rates and unit costs to be achieved.

The blind hole boring programme is also presented – it is expected that this type of equipment will soon become a normal production tool on the gold mines.

Raise-boring as carried out by Anglo American Corporation Technical Development Services is covered in some detail, cost and performance figures and problems encountered being given.

In conclusion there is a brief discussion of the problem areas that remain in hard rock boring, as well as those which require further research and development.

Introduction

DEEP-LEVEL GOLD MINING (FIG. 1)

South African gold mines of the Anglo American Corporation (AAC), of which there are nine, vary in depth from 430 to 3400 m below surface. The average working depth of most mines is in the region of 1700 m, the virgin rock temperature at that depth being 43°C. The gold-bearing reefs dip at about 20° and the average stoping width is of the order of 118 cm. The channel width of the reef does not often exceed 45 cm.

The various reefs occur in the rocks of the Witwatersrand System and, for the most part, the development and stoping excavations are situated in tough, highly abrasive quartzites, which have a uniaxial compressive strength of about 207 MPa (30 000 lb/in^2).

Extensive faulting of the gold-bearing reef has taken place and, consequently, mining layouts are complicated. They must, however, be sufficiently flexible to cater for sudden breaks in reef continuity. The size of footwall development excavations must be kept as small as possible to minimize support costs at these mining depths. Other adverse factors that are encountered on the gold mines are high humidity, the presence of methane and the harmful effects of fine dust with a high silica content. These factors necessitate heavy expenditure on ventilation and underground refrigeration facilities.

Elandsrand Gold Mining Co., Ltd., is a typical new deep-level Anglo American gold mine, where it is planned to hoist 280 000 tons of rock monthly from a depth of 2500 m. The quantity of air in circulation will be 750 m^3 sec (1 590 000 ft^3 min), with an installed refrigeration capacity of 60 MW

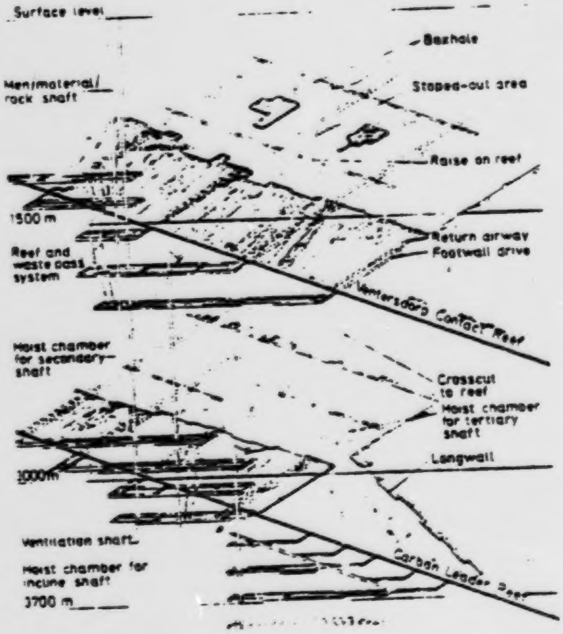


Fig. 1 Layout of typical deep-level gold mine

MECHANIZATION

Until five years ago labour was always readily available, which allowed South African underground gold mining to be labour-intensive. Productivity of the underground labour force is of the order of only 1 tonne per man-shift. As mining depths increase and environmental conditions become more arduous, manual labour becomes increasingly unpopular and unproductive.

In recent years the wages of underground workers in the South African gold mines have risen substantially, and this has contributed to an alarming escalation in working costs. It has, therefore, become imperative to contain cost increases by improving the productivity of the underground labour force by mechanization – indeed, it is felt that increased mechanization will ensure the long-term viability of the mines and that the generation of new technology and techniques could well render exploitable vast tonnages of low-grade ore that are currently unpayable.

The unique geological and mining conditions that are encountered in South African gold mines dictate that only some of the mechanization technology that has been developed elsewhere can be applied locally. In particular, the technology of mechanizing stoping operations will have to be developed locally, and this makes the task of instituting technological innovation that much more urgent. Accordingly, the South African gold mining groups and the Chamber of Mines of South Africa are spending large sums of money on the mechanization of underground operations.

HARD ROCK BORING

Currently, hard rock boring is one of the main activities in the Technical Development Services (TDS) mechanization programme. The objectives are to speed up the development that provides access to the orebody and to rapidly improve facilities for the transfer of men, ventilation, rock and materials, both adjacent to main shaft systems and to the stoping operations.

The number of raise borers that are employed in the division's mines has risen from one in 1973 to eight in 1977, one of the borers, the Robbins 85R, being among the largest raise borers to be operated underground.

At present (late 1977) three 1.5-m blind hole borers are under test; early in 1978 a new-generation blind hole borer, the Calweld BH.60 SP, which is capable of drilling 2.1-m holes, will begin trials.

Trials with a tunnel borer were begun in October, 1976, and, after extensive alterations, it was recommissioned in October, 1977. A second tunnel borer, which is currently being assembled, is scheduled to begin operations in January, 1978.

Raise-boring is now fully accepted by the mines, and most ore- and waste-passes, internal lift shafts and ventilation shafts are developed with raise borers. Four are being used concurrently with sinking operations in new shafts: when sinking and equipping of the shafts have been completed, the ore- and waste-pass system will have been completed at the same time, thereby speeding up development operations from the shaft position to enable the mine to generate a positive cash flow earlier than would have been possible with other raising techniques.

The trends in hole diameter required by the mines have progressed from 1.8-m diameter holes to, predominantly, 2.4-m diameter holes, some 3.6-m auxiliary shafts being required. The earlier 1.8-m holes tended to hang up when used as rock passes, particularly when the inclination exceeded 55°. Currently, most rock passes are bored to a diameter of 2.4 m.

Recently, the TDS 85R machine successfully bored some stages of a 3.6-m diameter lift shaft in medium hard quartzite, and the hydraulic drive of the machine enabled disc cutters to be used effectively, thus keeping cutting costs to a minimum; penetration rates in excess of 1.0 m/h were achieved.

The raise borers used by TDS have progressed from the earlier ac drive machines operated with fixed-speed gearboxes to hydraulic and dc electric drive machines. The latter two types of machine have greatly enhanced the versatility of raise boring, particularly in highly stressed and broken ground. The main advantage has been the completely variable rotation speed and torque characteristics that have enabled the machines to deal with the most difficult types of rock without undue vibration and strain to the drill string and to the machine frame.

Blind hole boring is in the final evaluation stage and it is expected that it will be introduced on a production basis into the gold mines by mid-1978. The main feature of this type of machine is that it requires only one access point to drill the hole – hence the term blind hole borer. No machine is, however, currently available for drilling holes larger than 1.5 m in diameter.

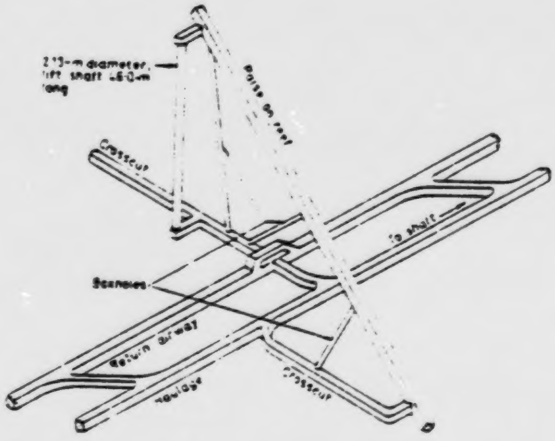


Fig. 2 Footwall development in gold mine showing boxholes and proposed lift shafts

The main application of the blind hole borer in the AAC gold mines is for the establishment of boxholes (small rock passes) from the footwall crosscuts to the stopes, some 20–30 m in length (Fig. 2).

The AAC gold mines develop approximately 58 000 m of boxholes per annum, most of this development being by slow and hazardous hand drilling methods.

With the imaginative use of mechanized blind hole and raise-boring machines it will be feasible to vary mine layouts more effectively than was possible in the past. One example is the increase in the level interval on mines, thereby effecting considerable cost savings and other advantages. It is envisaged that large numbers of blind hole borers will be required in the AAC gold mines.

With regard to tunnel boring, these machines are required to effect inter-shaft connexions and to push development headings to major ore-reserve areas as rapidly as possible. Such tunnels are envisaged to be between 1000 and 5000 m long.

Most tunnel-boring machines are specially designed for particular tunnels, and the first of the AAC tunnel borers has yet to be fully adapted to achieve acceptable advance and economic targets.

Blind hole boring

GENERAL

After the successful application of raise borers in the AAC gold mines, TDS embarked on trials with blind hole borers. These machines, unlike raise borers, do not require the establishment of both upper and lower positions before a hole can be bored and, therefore, they have a major application for the mining of boxholes to link footwall excavations with the reef horizons (Fig. 2).

Another important application will be the boring of lift shafts from the haulage to the stope horizon. The lift shafts will be equipped with elevators and used to rapidly transport men and materials into the stope. The boxholes envisaged will be mainly between 20 and 80 m in length.

Blind hole boring in AAC mines is still under development and, initially, trials were conducted with the main aim of determining the feasibility of boring boxholes by use of a full-face technique. As experience was gained, so the design of the equipment was altered and, recently, an order was placed for a third-generation borer with the facility to extract dust- and methane-laden air from the head area down through the centre of the drill string. The air is filtered and then returned to the mine through-ventilation system.

In the trials to date operational costs have been overshadowed by developmental costs, and performances have been affected by alteration delays. The potential of blind hole boring has, however,

Table 1 Holes drilled by TDS with 1.52-m (5-ft) diameter blind hole borers

		Location			Hole particulars			Geology	
Machine type	Hole no.	Mine	Shaft	Level	Length, m	Diameter, m	Dip angle	Rock type*	Distance, m
BH.80	1	Freddies Consolidated Mines, Ltd.	3	53	31	1.52	90°	U.F.1	31
BH.80	2†		1	56	122	1.52	61°	M.B.Q. U.S.M.	97 25
UR.60	1	Welkom Gold Mining Co., Ltd.	1	40	73	1.52	55°	U.F.1	73
UR.60	2‡		2	49	90	1.52	60°	U.F.1	90
UR.60	3		2	49	82	1.52	61°	U.F.1	82
BH.60	1§	Western Holdings, Ltd.	2	45	63	1.52	55°	U.F.2 Broken dyke	46 17
BH.60	2		2	38	52	1.52	55°	U.F.1	52
BH.60	3		4	45	49	1.52	80°	U.F.1 U.F.2	25 24
BH.60	4		4	53	22	1.52	62°	U.F.1	22
BH.60	5		4	53	36	1.52	70°	U.F.1	36
BH.60	6		4	53	65	1.52	61°	U.F.1 U.F.2	59 6
BH.60	7		3	33	30	1.52	60°	U.F.2 U.F.1–Z4 U.F.1–Z3 U.F.1–Z2 U.F.1–Z1	6 6.5 28.5 15 9
BH.60	8		3	28	65	1.52	65°	U.F.2 U.F.1	28 36

* Approximate uniaxial compressive strengths
Upper Footwall Quartzities = U.F.1 = 159–265 MPa (23 000–38 500 lb/in²)
Upper Footwall Quartzities = U.F.2 = 152–343 MPa (22 000–49 700 lb/in²)
Main Bird Quartzites = M.B.Q. = 62–193 MPa (9000–28 000 lb/in²)
Upper Shale Marker = U.S.M. = 69–121 MPa (10 000–17 500 lb/in²)
† Drill string broke at 95 m.
‡ Drill string broke at 75 m.
§ Drill string broke at 47 m.

Table 2

	Machine type		
	BH.80	**UR.60**	**BH.60**
Manufacturer	**Calweld (U.S.A.)**	**Subterranean (U.S.A.)**	**Calweld (U.S.A.)**
Nominal diameter, m (ft)	1.52 (5)	1.52 (5)	1.52 (5)
Dip angle adjustment	55–90°	60–90°	50–90°
Reaming thrust, kg (lb)	174 633 (385 000)	136 077 (300 000)	174 180 (384 000)
Reaming torque, Nm (ft lb)	108 465 (80 000)	51 521 (38 000)	81 349 (60 000)
Total installed power, kW (hp)	187 (250)	112 (150)	149 (200)
Type of drive	Electro-hydraulic	Electro-hydraulic	Electro-hydraulic
Cutter head rev/min	0–35	0–13.5	0–14
Drill rod length, m (ft)	1.22 (4)	0.61 (2)	0.61 (2)
Dimensions of assembled machine			
Height, m	3.96	3.30	2.21
Width, m	2.54	2.13	1.52
Length, m	4.17	3.43	2.34

been established and it is felt that the operating costs and performances that have been achieved by the latest Calweld BH.60 borer are reasonably representative of blind hole boring in the Witwatersrand System quartzites.

Table 1 summarizes the holes drilled to date, and Table 2 gives the general specifications of the blind hole borers that were used during the boring trials.

TRIALS WITH CALWELD BH.80 BLIND HOLE BORER (FIG. 3)

The Calweld BH.80 machine was introduced in 1974 and was used to drill two holes at Freddies Consolidated Mines, Ltd. The cutter head had a flat cutting profile and was dressed with ten Smith Boart button cutters with two conical strawberry-type cutters in the centre. The machine was too large for underground use in a South African gold mine.

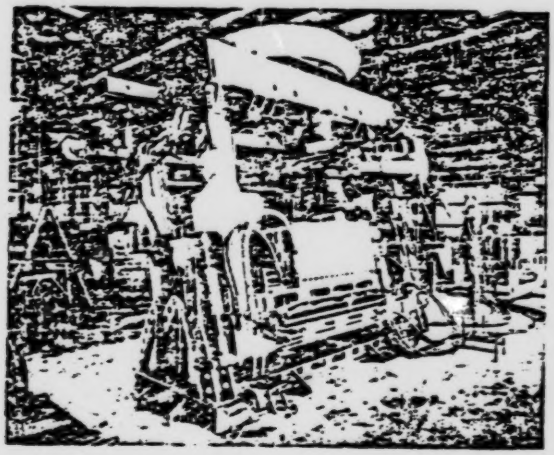


Fig. 3 Calweld BH.80 blind hole borer

TRIALS WITH SUBTERRANEAN UR.60 BLIND HOLE BORER (FIG. 4)

The Subterranean UR.60 machine was purchased by TDS in 1975 and was used to drill three holes at Welkom gold mine. The cutter head was similar to that of the Calweld BH.80 but instead of the two centre strawberry-type cutters a 'stinger' was instal-

Fig. 4 Subterranean UR.60 blind hole borer at Welkom gold mine

led on the top of the cutter head to assist in head collaring operations. The head was dressed with ten Kennametal button cutters with four stabilizing rollers.

TRIALS WITH CALWELD BH.60 BLIND HOLE BORER

From experience that had been gained from the BH.80 trials various refinements had been incorporated into the design and construction of the BH.60 to make it more suitable for use in South African gold mines.

The machine is considerably smaller than the BH.80 and wheels have been fitted to the main frame to facilitate transportation. The cutter head is transported on a car, which can lift the head to the top of the machine head frame.

The reamer head is similar to that of the BH.80 and was designed to use ten Smith-type button cutters with two strawberry-type cutters in the centre.

The machine was used to drill eight holes (see Table 1).

STABILIZER PROBLEMS AND DEVELOPMENT

The first two holes that were drilled with the Calweld BH.80 borer used rotating-type stabilizers. These were not satisfactory and were replaced by non-rotating type stabilizers during the drilling of the second hole. Problems were still experienced, however, the drill string breaking in two places and the hole deflecting off line.

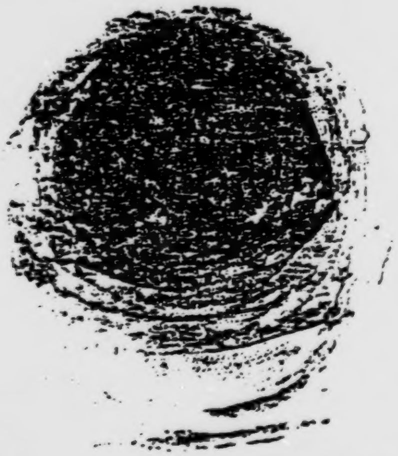


Fig. 5 Side wall flaking in 1.52-m boxhole bored with blind hole borer

The UR.60 machine used non-rotating stabilizers with six equally spaced radial arms. Problems were experienced with these arms, which broke off, and on several occasions the drill string had to be withdrawn from the hole to repair the stabilizers. During the drilling of the third hole two rotating-type stabilizers with four equally spaced outer rollers were fitted in addition to two non-rotating stabilizers.

Non-rotating stabilizers have been used with the BH.60 machine, but problems were experienced in fractured ground.

It was decided to substitute the rollers on certain stabilizers with skids to improve stabilization in fractured ground. The skids were 1.5 m long and were attached to the arms of two consecutive non-rotating stabilizers. Two of these skid-type stabilizers were used and the string stabilizers were spaced at 5.49-m intervals. Fig. 5 shows the cavities that result from side wall flaking in a 1.52-m diameter hole. It was for this type of condition that the skid type of stabilizer was designed. Fig. 6 shows a non-rotating skid-type stabilizer being installed.

Fig. 6 Skid-type stabilizer for drill string of 1.52-m Calweld BH.60 blind hole borer

Holes 4–8 were completed successfully, the reamer head remaining on line and holing through on target. The stabilizer configuration was similar to that used during hole 3.

ANALYSIS OF RESULTS

Table 3 shows the operating costs and performance figures for the BH.60 blind hole borer during the eight holes that were drilled. Inter-hole delays have been excluded.

Labour costs include an engineering foreman, two artisan operators and four assistants for double-shift operations, and include all salaries, bonus payments and overhead costs that are directly associated with the crew. Drill-site excavation and preparation costs and the capital amortization costs of the equipment have been excluded. Maintenance costs are those which were incurred from the time the borer was originally commissioned up until the time hole no. 8 was completed. It should be noted that maintenance costs include developmental costs. The cutters and reamer head were rented from the manufacturer at a fixed cost per metre.

The actual boring time excludes the time that was spent on normal rod handling and the installation of stabilizers. The instantaneous penetration rate is the number of metres bored divided by the actual boring time.

CONCLUDING REMARKS ON BLIND HOLE BORING TRIALS

Most of the major problems that are associated

Table 3 Summary of results achieved with BH.60 blind hole borer*

Hole no.	Metres bored	Hole diameter, m	Hole angle	Actual boring time, h	Instantaneous penetration rate, m/h
1	47	1.52	55°	50	0.94
2	52	1.52	55°	77	0.68
3	49	1.52	50°	50	0.98
4	22	1.52	62°	21	1.05
5	36	1.52	70°	51	0.71
6	65	1.52	61°	65	1.00
7	28	1.52	60°	32	0.87
8	64	1.52	65°	80	0.97

* Reamer hire = R 20.00/m
Cutter costs = R 74.90/m
Maintenance costs = R 65.44/m
Labour costs = R 70.53/m
Total = R230.87/m

Labour costs calculated from date hole was collared to date when reaming was completed. Cutter costs and reamer hire fixed at contracted rate per metre for duration of trials.

with blind hole boring have now been identified, and the method has proved to be a viable means of mechanically excavating boxholes. It is slightly more expensive than most conventional boxhole development methods, but it is faster and less hazardous.

The size of the machines currently available permits them to be handled and transported easily through shafts and haulages in South African gold mines.

The trend in hole size is towards 1.83- and 2.13-m diameter holes.

Boring trials have indicated the importance of adequate drill string stabilization and a well balanced adequately supported cutter head. Furthermore, it is important to provide adequate support for the cutter head during collaring.

A 'stinger' with a pilot bit on top of the cutter head allows for easy collaring and assists in stabilizing the cutter head.

Because of the ever-present danger in the South African gold mines of intersecting faults that contain methane during the boring operation it is necessary for an exhaust ventilation system to be incorporated into the machine to enable air to be circulated to and from the cutting face.

It is desirable that the machine base frame should incorporate a swivel mechanism to allow the borer to rotate on its base plate. This will enable the machine to drill two inclined boxholes in opposite directions from the same location.

Although this description has concentrated on blind hole borers with rotating-type drill strings, not all machines on the market are of this type. Another concept that is widely used in blind hole boring trials in South Africa is that of a non-rotating drill string.

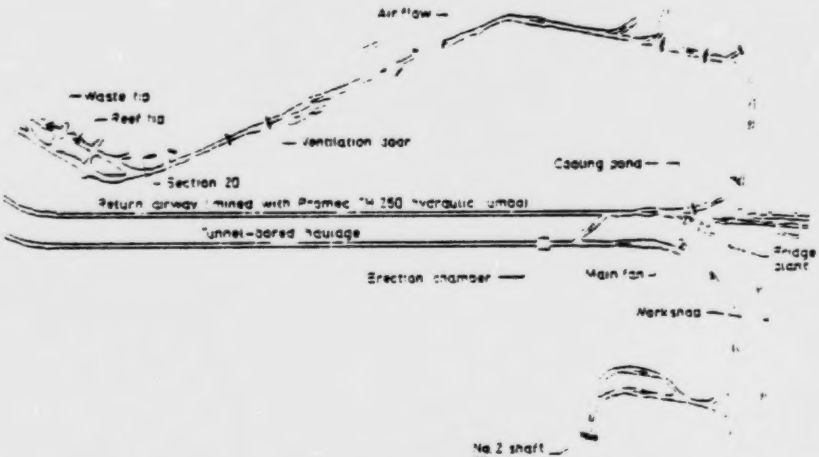


Fig. 7 Wirth tunnel borer – Free State Geduld Mines, Ltd., no. 2 shaft, 2/53 no. 2 haulage east (not to scale)

Tunnel boring

The AAC Gold and Uranium Division ordered a tunnel borer from Alfred Wirth and Co., West Germany, which was put into service in October, 1976. A second unit was ordered from Subterranean Tools, Inc., U.S.A., which incorporates several adjustments that were found necessary from testing the Wirth machine, and this is being

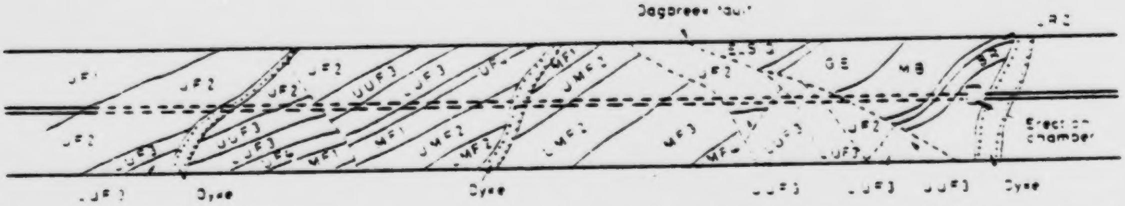


Fig. 8 Section, looking south, showing proposed 2/53 no. 1 and no. 2 haulage east (not to scale): ELS.Q., Elsburg Quartzites (siliceous to slightly drab quartzites); G.E., Gold Estates (drab yellow quartzites with polymictic conglomerate bands) M.B., Main Bird (drab yellow quartzites with argillaceous phases); L.R.Z., Leader Reef Zone (alternating grey siliceous and dirty yellow quartzites with polymictic conglomerates); U.F.1, Upper Footwall 1 (alternating yellow and grey siliceous quartzites); U.F.2, Upper Footwall 2 (yellow quartzites with polymictic conglomerate bands); U.F.4, Upper Footwall 4 (yellow quartzites with relatively polymictic conglomerates with basal oligomictic conglomerates); U.U.F.3, Upper Upper Footwall 3 (laminated grey siliceous quartzites with oligomictic grit bands); L.U.F.3, Lower Upper Footwall 3 (drab yellow quartzites with polymictic pebble bands); M.F.1, Middle Footwall 1 (drab yellow quartzites with loosely packed polymictic conglomerates); M.F.3, Middle Footwall 3 (yellow quartzites with argillaceous phases); M.F.4, Middle Footwall 4 (laminated grey siliceous quartzites); U.M.F.2, Upper Middle Footwall 2 (drab yellow quartzites with scattered polymictic grits); L.M.F.2, Lower Middle Footwall 2 (grey siliceous quartzites); B.R., Basal Reef (grey siliceous quartzites with oligomictic conglomerates)

erected on surface at the time of writing.

The Wirth machine is boring a 1.5-km tunnel on 53 level at Free State Geduld Mines, Ltd., no. 2 shaft. It was planned for the haulage to intersect the major Dagbreek fault, which has a throw of 488 m and where the ground conditions were known to be poor. Figs. 7 and 8 show a plan and geological section of the area where the tunnel is situated.

WIRTH TUNNEL BORER

The Wirth TB 11-340 H/Sch machine was designed by the manufacturers in liaison with TDS and the AAC Gold Division consultants (see Fig. 9).

Fig. 9 Wirth tunnel borer during assembly

The total cost for the tunnel borer, its back-up equipment and ancillary equipment, such as refrigeration plant and ventilation equipment, was R1 300 000.

The machine is designed to bore a 3.4-m diameter tunnel, in quartzite, at advance rates up to 1.5 m/h to give a daily advance of 18 m.

The basic borer design is the same as that used by Wirth on previous tunnel borers, but the ventilation and dust-suppression systems were designed and fitted locally. A technical description of the machine is given in Appendix 1.

UNDERGROUND ERECTION OF WIRTH TUNNEL BORER

All machine components were transported to the erection chamber in assembly sequences and the handling of the heavy outer and inner kelly was effected by use of a 15-ton crawl inside the chamber. Major erection was completed in ten days on a double-shift basis and a further two weeks were required to complete all power, hydraulic and service connexions. A final week was required for test running, pressure setting and 'walking' the machine into the start tunnel (Fig. 10).

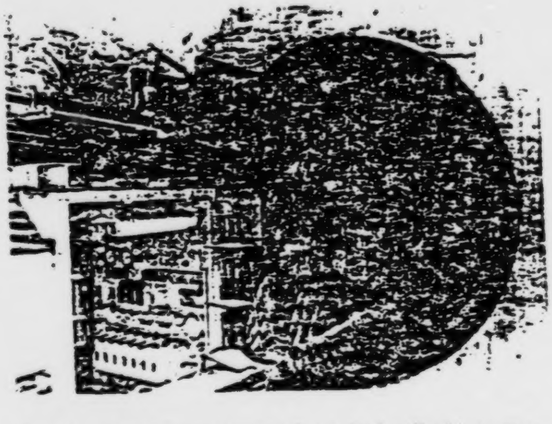


Fig. 10 Wirth tunnel borer after it had walked into start tube

VENTILATION AND REFRIGERATION OF WIRTH TUNNEL BORER

Revised ventilation plan

Difficulties were experienced in maintaining the machine on grade during the initial boring — this resulted in the clearance between the hoppers and the 1.0-m diameter column being reduced. Suspension problems on this column contributed to the clearance problem, and the column was damaged by hoppers at several points. The ducting also proved bulky to handle and install.

The design of the ventilation system was reviewed and it was decided to replace the large 1.0-m diameter column by a 0.76-m diameter column, which will result in the air quantity delivered in the machine area at holing being decreased from 12 to 7 m³/sec. This will reduce the heat that is removed by the air to 320 kW, representing a loss of 220 kW of cooling power. The loss will be offset by a second refrigeration machine, which was installed as a standby. Both refrigeration machines will be required to be run continuously when the tunnel is close to holing.

Refrigeration
Two Carrier 240-kW refrigeration machines are provided to chill all the service water that is sent into the tunnel. Each plant is capable of chilling 5 l/sec of water from 26 to 15°C. Summer inlet water temperatures to the plant have been as high as 29°C.

Currently, one machine is operated at a time with the second machine as standby.

Service water supply (Fig. 11)
Treated water is fed from the 49 level drinking water dam, which provides a head of 122 m, through a pressure-reducing valve to the refrigeration plant as well as to the spray pond to replace water that is lost by evaporation and carryover.

Normal usage of service water in the tunnel is approximately 5 l/sec. The water is chilled to approximately 14°C and enters the tunnel through a 100-mm diameter insulated pipe. A cooling coil is installed at the erection chamber to pre-cool air that enters the tunnel. The chilled water is used to cool two hydraulic power packs and for the two rockbolting machines. The remaining water is used for spraying at the head for dust-control and cutter-cooling purposes. Sprays are provided at the conveyor discharge points.

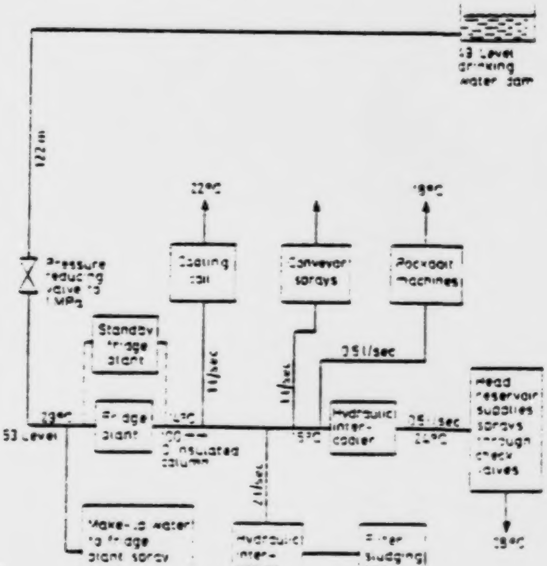


Fig. 11 Water circuit for Wirth tunnel borer

Head sprays
Redesigned head sprays that incorporate check valves and sieves have been installed to give better spray coverage of the face, reduce water accumulations at the head and to prevent rust and pipe scale from blocking the sprays.

Drainage
All the service water, with the exception of that which is used to cool the main hydraulic power pack, is dumped to waste and used to sludge the dust that is removed in the filter unit. Excess water is pumped out of the tunnel through a 100-mm diameter drainage column.

Force ventilation system
The main force ventilation system blows fresh air from the start of the tunnel to the machine area. The fan used is a Fantecnic PZF 137 centrifugal inlet guide vane controlled fan, handling 10 m³/sec of air through a 760-mm diameter flexible ducting.

A second force column is mounted on the borer power train to take fresh air to the operator's cab and the tracklaying area. Regulated openings in the sides of the column enable the air distribution pattern in the machine area to be adjusted easily.

During the summer months the inlet air to the tunnel had a temperature of 23°C wet bulb and 33°C dry bulb. Despite this, the temperature in the operator's cab during boring did not exceed 29°C wet bulb, even before the installation of the cooling coil.

It is possible for the borer to advance 10 m before installations of additional lengths of the main ventilation column become necessary. An expansion magazine is provided to cater for this advance and to connect the main ventilation column to the machine force column at all times.

An 11-kW fan, handling 4 m³/sec, is installed on the machine side of the magazine to pressurize the force column. The air that is released at the expansion magazine is used to cool the main transformers, which are installed at the end of the power train.

Exhaust ventilation system
The machine exhaust system draws 3 m³/sec of air from the borer head area through two 250-mm diameter fibre glass ducts mounted on each side of the conveyor. The fibre glass ducting is connected into a single exhaust type flexible column behind the operator's cab. The single column, which is 405 mm in diameter, carries the dust-laden air to the filter. The air velocity in the column is 23 m/sec

to prevent dust from settling out.

Filter unit

The dust filter is a Mikropul minipulsaire, self-cleaning, fabric filter. The filter chamber gives a total filtration area of 30.6 m^2 with a filtration speed of 0.10 m/sec on the bags.

Cleaning is by compressed-air reversed pulse. The dust is pulsed from the outside of the bags on to the floor of the filter chamber and removed by a screw conveyor to a mixing tank under the filter. It is sludged with water in the tank and then pumped out of the tunnel.

The pressures across the filter bags have been between 500 and 700 Pa during boring with an 8-sec pulse setting.

Dust counts in the tunnel during boring have been satisfactory, typical values (particles per millilitre) being operator's cab, 102; machine rear grippers, 109; Mikropul outlet (undiluted), 180; loading point, 112; main fan inlet, 41; tracklaying area, 54; and return air, 99.

OPERATIONAL BORING PROBLEMS

Large rocks and machine conveyor system

The machine commenced operations on 4 October, 1976, and was shut down for major alterations on 1 May, 1977, after 196 m of boring had been completed. During this period it became clear that the conveyor system was not suitable for handling fine wetted material, and excessive spillage had occurred at transfer points. Another problem was that the tunnel face tended to come away in relatively large chunks (some 0.6 m x 0.4 m x 0.3 m), which caused the head to jam and also damaged the front transfer chute continually. Moreover, because the conveyor system was unable to handle large rocks, a man had to be placed at each transfer point to position rocks safely on the belts.

The problems with the conveying system stemmed mainly from the specification given to the manufacturers that the machine had to operate on a 30-m radius bend. It had been the intention to mine crosscuts off main haulages with the tunnel borer. To bore around sharp bends the machine was fitted with five short conveyor belts instead of the one long belt that would normally have been used. The overlap from one belt to another was insufficient and this resulted in a build-up of fine sludge on the floor of the tunnel.

Cutter problems

Although the quartzite formations to be traversed are known to be relatively easy to bore, it was decided to bore, initially, with Wirth button cutters rather than the cheaper disc cutters. As it turned out, the gauge button cutters failed prematurely and had to be replaced hurriedly by cutters that were made with a denser button configuration. Furthermore, the wear of the tungsten carbide buttons on several cutters was not uniform. Experimentation with disc cutters is now taking place.

Weak ground

As can be seen from Fig. 8, the 53 level haulage runs through a major fault. Bad ground conditions were expected and the plan was to bore rapidly and continuously and to support the weak ground in an area where it could be roofbolted properly – this position was where a rockbolting platform had been installed 22 m from the boring head.

Because of the slow boring progress, however, the unsupported tunnel between the head and the rockbolting platform deteriorated so badly in places that, on occasions, boring had to be stopped to install support, and this proved to be a difficult and slow operation.

OPERATIONAL PERFORMANCE AND COSTS FOR WIRTH TUNNEL BORER

Results prior to shutdown

At shutdown in May, 1977, the Wirth tunnel borer had only bored 196 m, because of the various problems encountered. For the first two months the machine operated on a single-shift basis and on a double-shift basis thereafter.

Prior to shutdown the best performance per shift for single shift was 4.1 m and for double shift was 6 m.

Because of the poor utilization the unit costs incurred were high and the ratio between labour and other costs was disproportionate. It is not intended to discuss costs at this stage because they would be misleading and unrepresentative of normal boring operations.

Recommissioning of Wirth tunnel borer

At the time of writing the borer is undergoing recommissioning trials following shutdown alterations. The conveyors have been altered and are performing satisfactorily, but the modified sludge-handling system has not yet been commissioned.

Because there is a likelihood of large rocks being produced by future boring operations the buckets of the head have been altered to prevent rocks larger than 0.4 m x 0.4 m x 0.17 m from being picked up. The rocks will thus be confined inside the head and will be crushed until they are small enough to be gathered by the buckets. This method may appear to be a crude means of overcoming the problem of large rocks inside the head, but it is expected that the borer torque limiting device will prevent damage to the cutters and buckets during boring.

The best performance during recommissioning – a 6-m advance in a single boring shift – points to the potential of the equipment.

Wirth steel disc cutters are being tried at present, but they have proved disappointing. One of the main problems is to obtain an equitable balance with steel ring life, and it would appear that the head design and cutter configuration are such that rings of certain disc cutters do more work than others (Fig. 12). The disc cutter shown in Fig. 12 indicates little wear on one ring, but the other ring, which has apparently not had tracking support

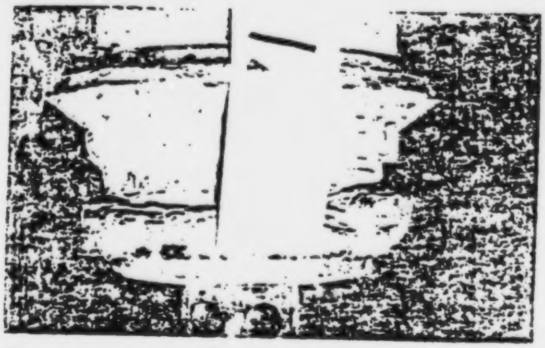


Fig. 12 Wirth disc cutters showing excessive damage to one ring because of inadequate tracking support from adjacent cutters

from the rings of other cutters, has collapsed completely.

It had been hoped that the use of disc cutters would result in a product that contained significantly less fines than that generated by button

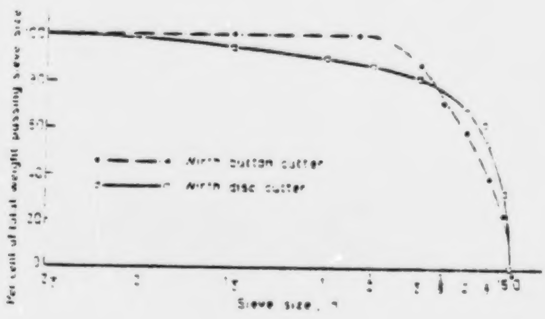


Fig. 13 Sieve analysis of Wirth tunnel-borer chippings for button and disc cutters

cutters. To some extent this has happened, but more fines have been produced by the disc cutters than was expected (Fig. 13).

STAFF AND LABOUR COMPLEMENT WIRTH TUNNEL BORER

The structure presented in Table 4 shows the complement for the Wirth tunnel borer for a 24-h operation with a target of 18 m/day.

SUBTERRANEAN TUNNEL BORER

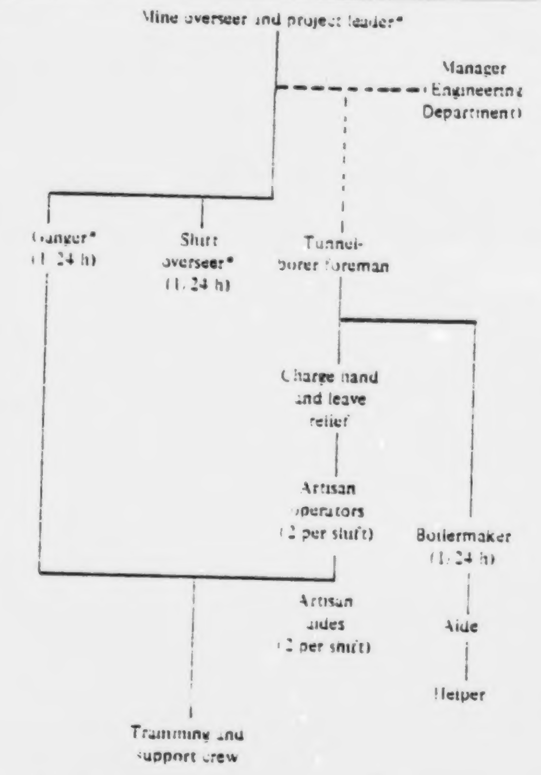
The Subterranean borer (see Appendix 2) is currently being assembled on surface; after testing it is scheduled to go underground in January, 1978 (Fig. 14). It is planned to bore a 2.4-km tunnel of 3.5-m diameter in quartzites at 1.5 m/h advance on 62 level at Vaal Reefs South gold mine.

The tunnel borer and its ancillary equipment cost R1 400 000.

Ventilation and refrigeration of Subterranean tunnel borer

Initial planning was based on 10 m/day, and the total estimated heat to be rejected from the tunnel

Table 4


Mine overseer and project leader*
Manager (Engineering Department)
Ganger* (1/24 h)
Shift overseer* (1/24 h)
Tunnel-borer foreman
Charge hand and leave relief
Artisan operators (2 per shift)
Boilermaker (1/24 h)
Artisan aides (2 per shift)
Aide
Helper
Tramming and support crew

Team leader (1/shift)
Loco driver (2/shift)
Loco guard (2/shift)
Roofbolting crew (2/shift)
Pipes, tracks and belt attendants (5/shift)

When normal boring operations are established this complement may be reduced

* Legal appointments in terms of Mines and Works Regulations.

is 1006 kW. The heat-removal capacity of the 30 m^3/sec of air provided in the tunnel during summer is estimated as 820 kW, and 400 kW of refrigeration can be made available from the mine refrigeration system. Thus, additional refrigeration is not required. A 37 kW E-type main force fan will deliver 10 m^3/sec of air to the head area through a 760-mm diameter column. Twin Bateman filter units will exhaust 4 m^3/sec from the head. It is planned that up to 4 l/sec of cooling water at 10°C

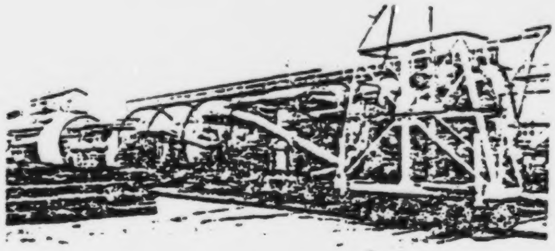


Fig. 14 Subterranean TB-II tunnel-boring machine during erection at TDS workshops

will be required for the head, borer and heat exchangers and to cool the incoming air.

Raise-boring

GENERAL

Raise-boring on AAC gold mines was started when TDS acquired a second-hand Robbins 61R machine in 1971. This machine indicated the economics of raise-boring in tough abrasive quartzite by achieving high rates of advance and increased safety. There are now eight raise borers in operation Robbins 61R, 71R, 72R, 83R and 85R machines and an Ingersoll Rand RBM7-SP machine.

In general, unit costs for raise-boring are still higher than those for conventional raising in the gold mines, but the main attraction of the borers is for the drilling of large-diameter raises at high rates of advance. In addition, raise-boring is a much safer technique.

Currently, all of the TDS raise borers are being used in sinking shafts, which enables the ore- and waste-passes to be mined concurrently with sinking operations. This is being done at Elandsrand, Free State Saaiplaas and President Steyn gold mines. Clearly, this procedure has tremendous financial attractions because it enables development from the shaft to commence earlier than would have been possible if the rock passes had been mined after the completion of sinking and equipping of the shaft.

The types of hole that are drilled by TDS borers are mainly 2.4-m diameter rock passes, and auxiliary ventilation shafts and 3.6-m diameter auxiliary lift shafts. The trend in hole diameter over the past six years has been from 1.8-m holes to, mainly, 2.4-m holes. The 1.8-m diameter rock passes caused hang-ups in the past, particularly at inclinations steeper than 55°, and are no longer drilled. Consequently, the 61R raise borer, which is virtually limited to holes of 1.8-m diameter, is almost redundant today.

Each TDS raise borer drills, on average, a 90-m hole each month. Current mine layouts are such that longer holes are rarely required. Nevertheless, although the short holes limit the utilization of the machines and, frequently, their full thrust potential is not applied, the torque capabilities of even the large machines are often taxed to the full. This is particularly so when kerf or disc cutters are used to drill large-diameter holes in tough highly stressed quartzite and lavas.

By adhering strictly to preventive maintenance schedules, coupled with major overhauls at intervals that do not exceed 18 months, the raise borers rarely break down during production boring. The mechanical availability of the machines is more than 95%, the calibre of the operators – mainly electricians and fitters – being an important contributory factor. The operators carry out major overhauls on their machines and become intimately knowledgeable of the equipment. Moreover, a competent operating and maintenance crew generates a high *esprit de corps*, which, in turn, leads to a low turnover of operators.

Raise-boring is now an established technique in many mines throughout the world and, therefore, it is not intended to discuss in detail the characteristics of the various machines that are on the market. Instead, the application of the various borers and cutter types in the TDS boring programme are covered and some general performance and cost data, as well as experience that relates specifically to TDS machines, are presented.

RAISE-BORING DATA FOR VARIOUS ROCK TYPES

To date, 106 raises, ranging in diameter from 1.2 to 3.6 m and varying in length from 33 to 182 m, have been drilled. The inclination of the holes has varied between 26° and vertical.

A total of 9500 m has been bored in various rock types of the Witwatersrand and Ventersdorp Systems. In the latter raises have been bored in tough, non-abrasive, basic ferromagnesian lavas with uniaxial compressive strengths that range from 241 MPa (35 000 lb/in²) to 413 MPa (60 000 lb/in²). In the lower Witwatersrand System holes have been drilled in abrasive quartzites that have ranged from tough and glassy to the medium-strength argillaceous variety. The uniaxial compressive strength of the quartzites bored varies from 138 MPa (20 000 lb/in²) to 276 MPa (40 000 lb/in²).

With the wide variation in hole diameter and borer, cutter and rock types, it is clearly not practicable to present condensed cost and performance information on TDS raise-boring. Accordingly, a selection of information that covers these variables is presented in Table 5.

SELECTION OF RAISE-BORING MACHINES

During the past two years there has been a dramatic increase in requests for raise-bored holes from the gold mines. Most of the holes that are drilled at present are 2.4 m in diameter, but the demand for 3.6-m diameter holes is steadily increasing.

This change in requirement, coupled with an advance in machine technology, has meant that TDS has not standardized on any one type of raise borer but has purchased its machines according to the hole requirements and latest machine type that was available when the purchase of a new machine was being considered.

Another factor that has influenced the selection of raise borers has been the state of the art with cutters. Cost escalation with tungsten carbide button roller cutters has been alarming in recent years, and it has become necessary to use disc cutters, formerly regarded as only suitable for soft rock, to bore medium to hard quartzite. In many medium-strength rocks disc cutters are superior to button cutters with regard to unit cost and penetration rate, but a disc head is generally not as smooth-running as a roller button cutter head, and it sets up severe vibration and torque fluctuations at the machine. It is this fact that militated against the use of disc cutters with the earlier

Table 5 AAC Gold and Uranium Division TDS general raise-boring data for quartzites of Witwatersrand System

Machine	Mine	Date	Hole diameter, m	Hole length, m	Rock type	Make and type of cutter	Average instantaneous penetration rate, m/h	Cutter cost, R/m*	Labour cost excluding delays, R/m	Maintenance costs, R/m†	Pilot bit, R/m	Total cost, R/m‡	Total cost, R/m³ ‡
Robbins 61R	P. Steyn	April, 1972	2.11	55	U.F. 1	Smith (button)	0.30	185.46	88.70	Average for period 1/72 7/72 41.40	22.04	337.60	202.96
	V. Reefs (N)	May, 1972	1.83	125	M.B.	Robbins (disc)	1.15	97.11	23.05		11.12	172.68	112.49
	P. Steyn	May, 1973	1.22	100	M.F. 2	Reed (button)	0.99	83.04	26.88	Average for period 2/73 - 9/73 44.60	12.51	166.03	124.70
Robbins 71R	W. D. Levels	June, 1974	2.13	110.0	M.B. glassy quartzites	Reed (button)	0.89	220.63	45.66	Average for period 3/74 11/74 46.28	15.28	327.85	167.67
	V. Reefs (S)	Oct., 1974	2.44	111.0	M.B.	Robbins (disc)	0.37	113.86	108.47		18.42	287.03	197.45
Robbins 72R	V. Reefs (S)	Aug., 1976	2.13	109.0	M.B.	Robbins (disc)	1.12	120.00	73.14	Average for period 3/76 – 11/76 51.37	23.12	267.63	180.51
	V. Reefs (S)	May, 1976	2.44	101.0	Dyke	Ing. Rand (button)	0.46	153.76	176.08		25.02	406.23	286.25
Robbins 83R	V. Reefs (S)	Oct., 1973	2.44	93.0	M.B. 6	Reed (button)	1.04	168.34	65.92	Average for periods 1/73- 3/74 22.67 1/76 - 7/76 56.76	24.15	281.14	148.69
	P. Steyn	April, 1976	2.44	65.0	U.F. 2	Wirth (button)	1.02	194.97	125.00		54.54	436.27	282.87
Robbins 85R	P. Steyn	April, 1977	2.44	77.0	M.F.	Robbins (disc)	0.96	165.00	90.38	Average for period 6/76 7/77 48.24	54.10	357.73	227.90
	V. Reefs (S)	Aug., 1976	3.66	87.0	M.B. 6	Robbins (disc)	0.61	254.51	110.61		44.66	458.02	227.54
Ingersoll Rand RBM7 SP	V. Reefs (W)	Jan., 1977	2.13	51.0	M.B.	Robbins (disc)	1.08	140.00	76.95	Average for period 6/76 - 2/77 22.64	9.44	249.03	147.39
	W. Holdings	Sept., 1976	2.13	60.0	U.F. 1	Ing. Rand (button)	1.02	184.69	71.02		41.96	320.31	186.22

* Includes costs for reamer head and cutter rental.

† Maintenance costs pro rated for meterage of each machine during given periods to distribute major spares costs.

‡ Total costs shown exclude site preparation, drill string and stabilizer repair, and labour charges during delay periods.

M.B. – Main Bird; U.F. – Upper Footwall; M.F. = Middle Footwall; P. Steyn – President Steyn; V. Reefs (S) – Vaal Reefs South; V. Reefs (N) = Vaal Reefs North; V. Reefs (W) – Vaal Reefs West; W. D. Levels – Western Deep Levels, W. Holdings – Western Holdings, Ing. Rand – Ingersoll Rand borers. which were driven by ac motors and had limited speed and torque control.

When the need to use disc cutters as a means of limiting the escalation of boring costs became imperative. coupled with the increasing demand for 3 6-m diameter holes. TDS decided to use hydraulic drive and dc electric drive borers when the demand for raise-bored holes necessitated the purchase of more machines.

It is appreciated that borers that are driven by an ac electric motor through a gearbox are simple and reliable machines and have an application where the rock types are consistent and homogeneous. but they do not suit TDS requirements.

The variable rotation speed and variable torque characteristics of the hydraulic drive and dc drive machines have enabled disc cutters to be used to a far greater extent than with the ac drive machines. and this. in turn. has kept unit costs for cutters to a minimum.

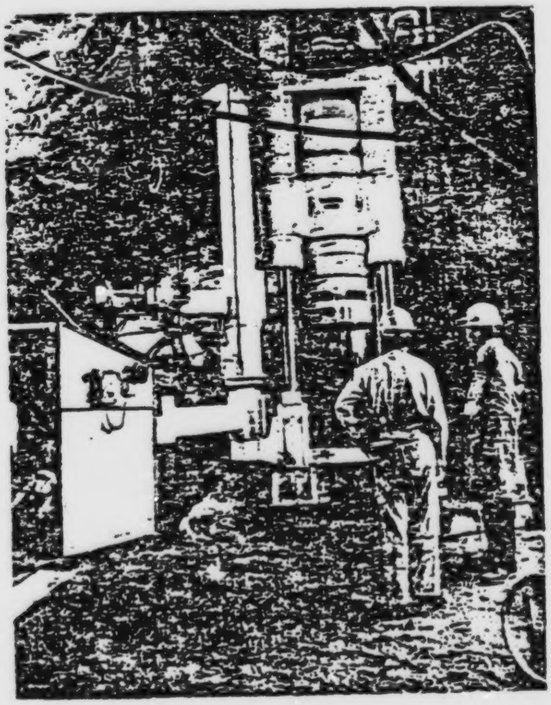


Fig. 15 Robbins 85R raise borer at President Steyn Gold Mining Co., Ltd.

In certain cases in hard quartzite disc heads require to be changed more often than button cutters. but the higher penetration rates and lower costs more than compensate for the extra downtime. TDS experience is that inter-site delays are far greater than those which are incurred by having to change disc cutters during the reaming of a hole.

The use of disc cutters with the Ingersoll Rand RBM7-SP (dc drive) and Robbins 72R (hydraulic drive) machines for the drilling of 2.4-m holes. and of the Robbins 85R (hydraulic drive) for the drilling of 3.6-m holes. has greatly enhanced the attraction of these machines.

Fig. 15 shows the Robbins 85R raise borer at President Steyn gold mine. and Fig. 16 a 3.6-m diameter domed head fitted with Robbins disc cutters in operation with the 85R raise borer.

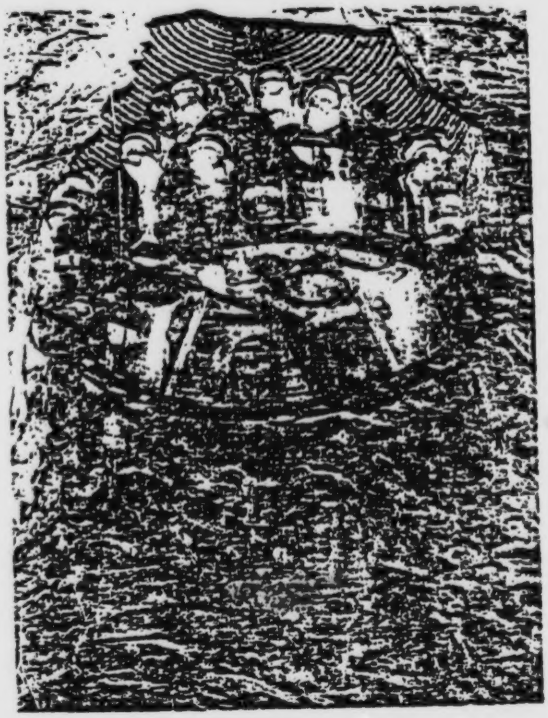


Fig. 16 Robbins 3.6-m domed head and disc cutters in use with 85R raise borer

RAISE-BORING POLICIES. TECHNIQUES AND PROBLEMS

Pilot-hole drilling

Clearly. in raise-boring it is imperative to maintain pilot-hole accuracy to prevent delays in locating holes that have missed their target. Moreover. holes that are required to be equipped as lift shafts must be drilled extremely accurately to accommodate guides for skips and conveyances.

When drilling pilot holes it is policy to limit penetration rates and to ensure that effective stabilization of drill-holes is practised by use of an adequate number of properly maintained stabilizers, which are as near as possible to the size of the pilot bits that are used. In addition. all raise borers are lined up by theodolite.

Before drilling is begun. collaring of the hole begins with the aid of the starter pipe and starter bush assembly and. initially. penetration rates do not exceed 0.6 m/h. After careful drilling of the first 1.5 m the roller bit sub is then installed on to the pilot bit and the starter pipe assembly is again used to continue collaring. Pilot drilling proper does not begin until the pilot bit. roller bit sub and three hexagonal stabilizers have been installed.

For 279-mm (11-in) diameter holes piloting rates are not permitted to exceed 3 m/h. and for 13¾-in holes a pilot drilling rate of 4.5 m/h is not exceeded. Moreover, the thrust on the pilot bit is not permitted to exceed 1051 kN/m (6000 lb/in) of pilot bit diameter.

This policy has resulted in hole deviations of less than 1% on hole length and, frequently, in holings taking place exactly on target.

Currently, only air flushing of pilot holes is used with TDS raise borers, but if long holes (in excess of 200 m) are ever required, water flushing will be carried out.

DRILL-ROD AND STABILIZER FAILURE

Two sizes of drill rod are used – 254-mm (10-in) rods with the RBM7-SP and 72R, and 327-mm (12⅞-in) rods with the 83R and 85R.

The larger-diameter components have not caused any problem, but in the past two years the failure of two 254-mm rods and the unsatisfactory life achieved with 279-mm hexagonal stabilizers and reamer stems have caused serious concern. The main failure with the stabilizers has been cracks that have developed in the box end and wrench flats.

Under investigation at present is the means of 'torquing in' reamer heads underground. The use of a special tool to make up and break out the connexion between the reamer head saver sub and the stabilizer at the bottom of the string has become imperative. This is, however, not necessarily the cause of 279-mm stabilizer problems, because failures have occurred with stabilizers that have been 'torqued in' with the borer at the top of the hole.

The drill-rod manufacturers are currently investigating the design of a make-up and break-out tool, which will enable joints that need to be made up remotely from the machine to be properly torqued in.

Until the cause of drill string and stabilizer failure has been clearly identified, the power output of all the TDS raise borers has been downgraded in accordance with the altered tension–torque diagrams issued by the drill-rod manufacturer. This has meant that the power output of TDS borers has been decreased by approximately 12%.

Because of the concern over component failures all drill string components undergo stringent ultrasonic and electromagnetic testing after each hole is completed. Latterly, this has been extended to every third hole for the 327-mm drill rods because no cracks have been observed in this type of rod in service.

Conclusions

Hard rock boring machines, particularly raise borers, have important applications in the AAC mechanization programme.

Raise borers will continue to be used for holes up to 250 m long, and it is expected that they will have an increasing application for the drilling of large-diameter holes.

Tunnel boring is relatively new and the machines have yet to be fully adapted to the conditions that are encountered on the gold mines.

The development of blind hole borers has progressed steadily in the past two years and, when the design of the stabilizers has been perfected, these machines will be ready for full production operations.

The vigorous efforts of both manufacturer and user are required to reduce unit costs and to improve machine performance. In particular, cutter costs must be singled out for special attention.

For its part, TDS is tackling the problem with various research establishments and is improving the instrumentation and data processing facilities on all its borers. The objective will be to relate rock-drillability characteristics to meaningful machine performance data – for example, thrust, torque and rotation speed. This information will then be linked up to unit costs and penetration rates for various rock, cutter and machine types.

It is considered that a breakthrough in performance with boring machines can be achieved by the use of high-pressure water jets at the cutting face, and TDS is currently pursuing this matter.

Acknowledgment

The authors acknowledge the assistance of several members of TDS staff in the preparation of this paper, and, in particular, W. J. van den Berg, Dr. A. E. Hall and P. W. Gorman.

Appendix 1

TECHNICAL DESCRIPTION – WIRTH TUNNEL BORER

Type – T'B S-II – 340 – H/Sch

The borer is mounted on a twin-track system, one track being used for muck removal and accessing material to the front of the machine, and the other for the power train of the borer. Both sets of tracks are 750-mm gauge.

The machine is powered through an electro-hydraulic system, giving full rotation and torque control. Horizontal and vertical steering is effected by hydraulic cylinders. Directional control is maintained by a laser beam, which strikes two targets 3 m apart near the front of the machine.

Rotary speed

0–15 rev/min, completely variable

Torque at cutter head

320 000 Nm at 22 MPa

Thrust

5300 kN

Feed

Stroke 800 mm
Speed 0–50 mm/min
No. of cylinders 4
Cylinder diameter 280 mm
Hydraulic pressure 22 MPa

Bracing

Force 15 000 kN
No. of shields 8
No. of cylinders 16
Cylinder diameter 200 mm
Hydraulic pressure 22 MPa

Electrical system

Total installed power 540 kW
Current Three-phase ac
Primary voltage 6.6 kV
Secondary voltage 380–500 V

Weight

Total 100 t
Cutter head 12 t

Cutters used

(1) 21 Wirth full-face button cutters
2 Wirth cone cutters
(2) 21 Wirth double disc cutters
2 Wirth cone cutters

Outer kelly

The outer kelly is of split design to reduce transport weight. It is fitted with eight bracing pads to ensure stability during boring, thereby limiting vibration and excessive cutter wear.

Inner kelly

The inner kelly is of square section to provide the necessary torque reaction against the outer kelly during boring. The inner kelly also carries the main bearings, drive shaft and gearbox; the gearbox is of the bull-ring planetary-gear type, powered by five Bruninghaus hydraulic motors.

Steering

Steering is carried out during the resetting cycle. The main outer kelly bracing pads are retracted and the vertical and horizontal steering cylinders are used to reposition the outer kelly.

Borer power train

The complete power train is made up of driver's cab, auxiliary power pack, rockbolt platform, conveyors, dust-extraction unit, main hydraulic power pack, electrical power pack, transformers and cable cars. The power train is connected to the borer by means of two hydraulic tubes, which are designed to sense any increase in the towing effort that is required to pull the train and to warn the operator of any obstruction.

Hydraulic power system

The hydraulic system is powered from three hydraulic power packs that control the following components: (*a*) *auxiliary system* – powered by one pack, and (*b*) *rotary cutting head power unit* – powered by the other two power packs.

Chilled water, which enters at 16°C, is fed from a refrigeration plant to cool the hydraulic power packs.

Conveyor system

Because of the short turning radius that was specified by TDS five conveyors were fitted to the borer, mounted above the borer and power train. The width of each conveyor is 500 mm and four are 10 m long and one is 12 m long. The conveyors are hydraulically driven and speed variation is possible.

Muck-removal system

Muck removal is effected by use of 4-t hoppers and 8-t battery hydrostatic locomotives. The hoppers are narrow, of double-side discharge type, with top overlap to permit continuous loading without spillage. Twenty hoppers are used, i.e. three spans of six with two spare.

Tracklaying

Tracklaying is carried out in the clear space beneath the no. 3 conveyor. Preformed concrete sleepers fit into the tunnel invert and carry four rails for the twin-track system. A mono-rail system is used to convey the sleepers and rails from the end of the permanent track to the tracklaying area.

Rockbolting system

A rockbolt platform is attached to the machine behind the auxiliary power pack some 22 m from the face. This platform carries four slide-mounted jackhammers on feed beams, which can drill 1.8-m rockbolt holes for temporary support.

Pumping system

In most tunnels the handling of water and sludge presents problems. In this particular tunnel the grade is only 1 in 500, which is insufficient for the water to run freely out of the tunnel. A pumping system has been designed to deal with water and sludge, although during boring operations a minimum of water is allowed to flow to the head for dust control.

Flameproofing

All electrical equipment on the borer has been granted a Government Mining Engineers' Certificate of Approval in terms of Mines and Works Regulation 21.16.

Methane detectors

G.T.M. methane detectors are used on the borer. The remote heads are mounted to sample the exhaust air from the dust-filtration unit. The instruments are positioned in the operator's cab and give an audible and visual signal if the methane level exceeds 1%. Monitoring is continuous throughout the shift.

Appendix 2

TECHNICAL DESCRIPTION – SUBTERRANEAN TUNNEL BORER

Type – PB-115

The borer is mounted on a 750-mm gauge, single-track system, and will utilize a California switch to permit shunting of muck and material cars behind the borer.

The machine is powered by an electro-hydraulic system, which gives full rotation and torque control. Horizontal and vertical steering is effected by hydraulically operated gripper shoes. Directional control of the tunnel will be maintained by a laser system.

Rotary speed

0–11 rev/min. completely variable

Maximum torque at cutter head

542 000 Nm

Thrust

4900 kN

Feed

Stroke 762 mm
Speed 0–50 mm/min
No. of cylinders 4
Cylinder diameter 250 mm
Hydraulic pressure 28 MPa

Bracing

Force 13 800 kN
No. of shields 10
No. of cylinders 2: 8
Cylinder diameters 203 and 254 mm
Hydraulic pressure 21 and 35 MPa

The main bracing is applied vertically because the tunnel side wall is expected to deteriorate significantly, which would restrict horizontal gripping.

Electrical system

Total installed power 746 kW
Current Three-phase, ac. 50 Hz
Primary voltage 6.6 kV
Secondary voltage 550 V

Weight

Total 150 t
Cutter head 17 t

Cutters used

2 Kennametal centre cone button cutters
6 Kennametal gauge button cutters
14 Kennametal double row disc cutters

Driving system

Eight 112-kW hydraulic motors drive the machine gearbox, which consists of double planetary reduction gears connecting the motors to the drive shaft by bull pinions and splined bull gear.

The drive shaft passes through the hollow backbone of the machine to the cutter head. The rotational speed of the drive shaft and cutter head is controlled by the operator during boring.

Steering

Spot movements of 50 mm may be carried out during the stroke of the machine, but, normally, steering is only carried out during the resetting cycle. Two independent gripper systems permit vertical and horizontal adjustment of the machine for maintenance of line and grade.

Borer train

The train comprises the rockbolt platform, operator's cab, auxiliary and main hydraulic power packs, dust-extraction unit, electrical power pack and workshop car. From the workshop car (44.7 m from head) to the end of the train there are seven gantry cars, 1.8 m wide by 2.2 m high, inside which the hoppers travel up to the tool car. The frame is of a steel rectangular I-beam construction, and has the fourth conveyor, services and ventilation ducting mounted above it.

Conveyor system

The four conveyors have a combined length of 70.0 m and a speed of 60 m/min. The conveyor belting is 460 mm wide, and each flight is driven by a 30-kW electric motor. The conveyor system enables bends of a minimum radius of 36.6 m to be negotiated.

Muck-removal system

Muck removal will be effected by use of specially designed Robert Hudson 5-t capacity hoppers and two 8-t Hunslet Taylor hydrostatic locomotives. The hoppers are narrow, of the double-side discharge type, with top overlaps to permit continuous loading without spillage. A total of 20 hoppers will be bought, which will provide three spans of six cars each with two spare cars.

Tracklaying

Temporary tracklaying will take place beneath the second conveyor flight, but permanent track installation will be done behind the California switch. A mono-rail has been installed through the centre of the back-up system to convey temporary rails and sleepers.

Rockbolting

The rockbolt platform will be situated in front of the operator's cab. The platform is tubular in construction and will carry four slide-mounted jackhammers on feed beams and will enable 1.8-m rockbolt holes to be drilled for temporary support.

Pumping

Pumping arrangements have not yet been finalized.

Flameproofing

All electrical equipment on the borer and its ancillaries will require a Government Mining Engineers Certificate of Approval in terms of the Mines and Works Regulation 21.16.

Methane detectors

G.T.M. methane detectors are to be used on the borer. Remote sampling heads will be sited in the exhaust air from the Bateman dust-filtration unit. The instruments will be placed in the operator's cab and will give warning signals for methane concentrations in excess of 1%.

Experiences gathered in drilling

an inclined gallery through hard rock

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Experiences gathered in drilling

an inclined gallery through hard rock

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For several years now the conventional methods applied in tunnel and gallery construction have been giving way to a host of fully automatic tunnelling machines. This development, which is a quite natural result of modern technology, relieves the miners of their dangerous job. Such notions as 'drilling – loading – blasting – ventilating – mucking' will very soon be things of the past. The crew will consist of mechanics and engineers, the working faces will resemble factories more than anything else. Everyone who is familiar with the extremely hard, dangerous and highly noxious job of the tunnelling miner will surely welcome this development. Already today it is difficult to recruit qualified miners since the interest in this trade has steeply declined over the years, which may be traced to the danger of silicosis. This tendency, with the rapid increase in wages, must necessarily lead to a search for new ways, special emphasis being attached to

1 small wage quota
2 high advance rates
3 safe working methods

The breathtaking development in the field of fully mechanical tunnel and gallery construction has advanced even in Europe, to the point where planning authorities consider first the employment of drilling machines. The sections completed to date have meanwhile reached a length of several kilometres. It is the hardness of the rock rather than the diameter that still presents certain difficulties. The high cutter cost and the heavy stress the machines are exposed to in rocks of more than 2,000kg/cm² compressive strength caused the failure of quite a number of test drillings. As a consequence, conventional methods were still given preference under such conditions.

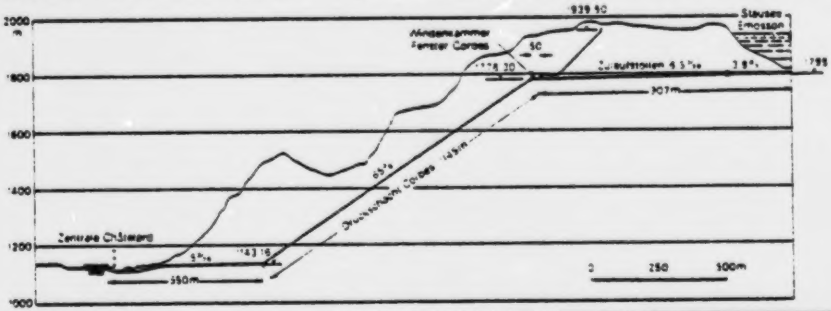
Test gallery

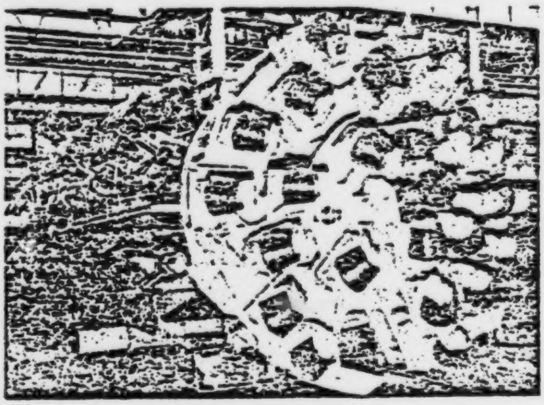
Even so, in the spring of 1968, Murer AG of Erstfeld (Switzerland), decided to try once again to drill a full-hole section in hard rock. It was made dependent on the outcome of the test gallery (compressive strength 2,400kg/cm² and high quartz content) whether a 1,140metre long gallery at 33° incline and with 3m dia through the same type of rock (Granit de Vallorcine) should be drilled or not. This inclined gallery was part of a contract for the Emosson power station project in the Valais Alps (Fig. 1).

The last 50m long section of a horizontal gallery (total length 350m) at the foot of the inclined gallery was selected for the test. The geological features were identical with those of the inclined gallery.

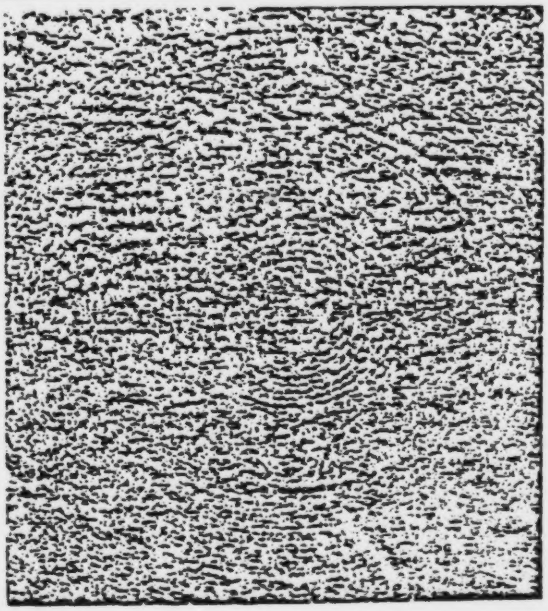
The test (2·40m dia) started on April 6, 1968. The machine used was a type TB I-240 Wirth tunnel drilling machine, whose drill head was equipped with 16 carbide insert cutters. The cuttings were taken by shovels fitted behind the drill head and

Fig. 1. Location diagram for inclined gallery.





Above: Fig. 2. The drill head of the Wirth machine Type TB II-300 had 26 carbide insert cutters, which roll on the face due to the rotation of the drill head.



Right: Fig. 3. The characteristic imprint of the cutters on the granite face.

dumped on a conveyor belt running over the machine. This test drilling turned out to be a total success: within a fortnight the 50m long section had been completed. So we had a proof of the drillability of the granite in question. Thanks to the experience gathered with the small machine we could proceed then with the preparations for drilling the inclined gallery ranging from 2·85 to 3·00m dia.

From the outset, we were fully aware of the difficulties we would have to face. The experiences gathered elsewhere were of no use, since we were the first to use a tunnel drilling machine for an inclined shaft and, what is more, in extremely hard rock. What mattered above all was thoroughly designed and well functioning follow-up equipment, since every breakdown would entail stopping the drilling machine.

The equipment selected from this point of view has already been discussed in detail elsewhere* and will be touched on only in passing here. The fact that the non-machine disturbances made up 3% of the overall worktime only may be regarded as proof that we were right.

Wirth machine begins drilling proper

The drilling operations proper were started on Oct. 1, 1968 with a machine, type TB II-300, of the same manufacturer (Wirth). The 'breakthrough' took place on Sept. 1, 1969 and met with lively interest from among experts.

The drill head was equipped with 26 carbide insert cutters (Fig. 2) which are noted for their cutting – or more aptly – breaking effect. The cutters – having no drive of their own – roll on the face due to the rotation of the drill head, and leave their characteristic imprints there (Fig. 3). The thrust makes the carbide inserts penetrate into the rock, whereas the break-off is caused by the rotation. The cuttings contain particles of all sizes, up to the size of a 5-Franc-piece, in addition to ultra-fine grains bound by the drilling mud. This material is particularly suitable for road construction and low-quality concrete (Fig. 4).

Ten metres of the above horizontal section were drilled with disc cutters for trial purposes. This test, however, proved to be a failure as the discs used at the time had worn off within a very short period, whereas the bearings showed no such symptoms. Nevertheless, there was one positive result in that the Wirth company was able to improve the disc cutters remarkably on the strength of the experience gathered. Another trial in the same rock some time later yielded considerably better results with harder discs.

It was planned to stagger the diameter of the penstock pipe in the pressure gallery. This enabled us to drill three different diameters, viz. 2·85–2·92–3·00m. True, there were two transition sections required; on the other hand we managed to save considerable cost for concrete. The reaming from the smaller diameter to the bigger one was done by the machine itself. An initial attempt to remove the 4cm with the help of drill hammers failed on account of the hardness of the rock. The drill head was prepared for the bigger diameter by displacing the six cutters of the outer row accordingly. This work in addition to the adaptation of the clamping shields and the height turned out to be somewhat expensive and troublesome. The cost incurred was, however, covered by the concrete cost saved. In future, however, it would be advisable to drill with one and the same diameter in similar cases. What mattered for us was the experience we were able to gather with the adaptability of the machine in terms of diameter.

Horizontal galleries still present certain difficulties (such as wear of chain conveyors, failure of conveyor belts) with regard to the transport of cuttings from the drill head to behind the machine: quite in contrast to inclined galleries! The cuttings are discharged here through an aperture in the drill head and removed with the drilling mud which cools the cutters. It is, however, indispensable to have some kind of guide which may take the form of a closed duct laid behind the machine to avoid the jump of stones if the machine hits on fissures. The enormous abrasive effect this flow of material has leads to the formation of a more or less deep channel which is by no means a disadvantage. With sufficient depth, there is even room for a tube which may serve for draining the mountain water, for receiving a cable and the like.

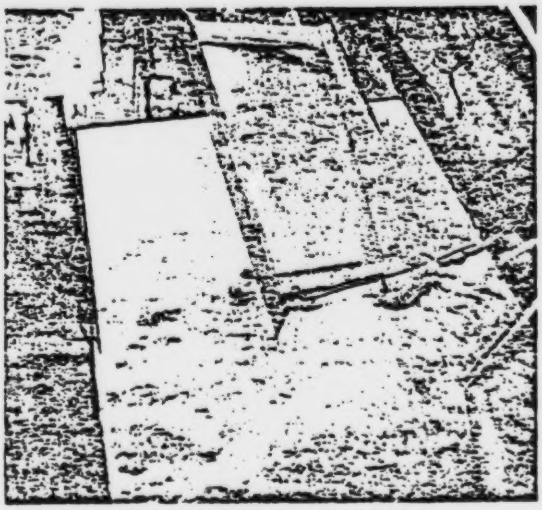


Fig. 4. Drilling mud, suitable for low-quality concrete.

The precondition of a smooth removal of cuttings is, of course, a sufficient angle of incline, the lowest limit being some 25°. If necessary, it is even possible to remain below this incline by adding an adequate water volume.

The machine was supplied with current through a 4,000 volt high-tension cable. Contrary to the commonest method, which is the installation of a stationary cable drum at the portal and the trailing of the cable, we opted for mounting the drum onto the machine proper. This procedure had certain assets: the cable to be reeled off is exposed to powerful tension forces and, as a result, rather expensive. This cable was but 200m long in our case. It was laid temporarily behind the machine, replaced by a simple high-tension cable and reeled again. There is neither the need of guide pulleys nor the

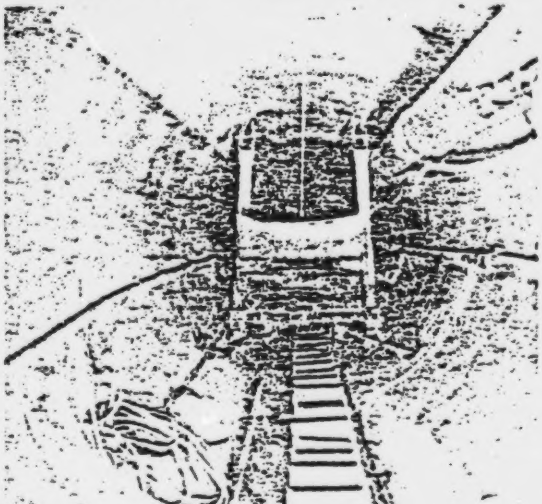


Fig. 5. The machine was adjusted by means of a laser beam.

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*Hambach, P.: Auffahren eines Schrägschachtes von 33° Steigung im Granit mittels einer Wirth-Tunnelbohrmaschine. *Der Tiefbau* 11 (1969) S.420/23, 437.

danger of the cable getting stuck on any obstacles. The power take-off at the cable drum was effected axially via sliding contacts. The cable was accommodated on a trailer along with a hinged oil transformer of 4,000/380 volts.

Guidance by laser (Fig. 5)

The machine was adjusted by means of a laser beam. To this end, there existed two sighting mechanisms fitted on the machine at a distance of some 7m (Fig. 6). The laser equipment was remounted at 300m intervals. It is not advisable to have longer distances with the devices and optical facilities existing at the present time, since the beam grows unsteady and too thick with increasing distances on account of the impure air in the gallery. It is, furthermore, recommended to check with conventional methods to have an additional control. A slight refraction of the laser beam which may result from water vapour or sudden water springs from the rock may very quickly lead to a deviation from the nominal axis. I need not dwell on the rather troublesome effects this can have, especially in bored galleries.

The ventilation was done by suction. An air-duct mounted behind the operator's stand of the machine exhausted the air from right behind the drill head by means of two plastic pipes of 200mm dia. flushing it then into the rigid ventilation line of 400mm dia. An axial fan at the portal served for removing the air. Measurements taken in the operator's stand by the Schweizerische Unfallversicherungsanstalt proved satisfactory.

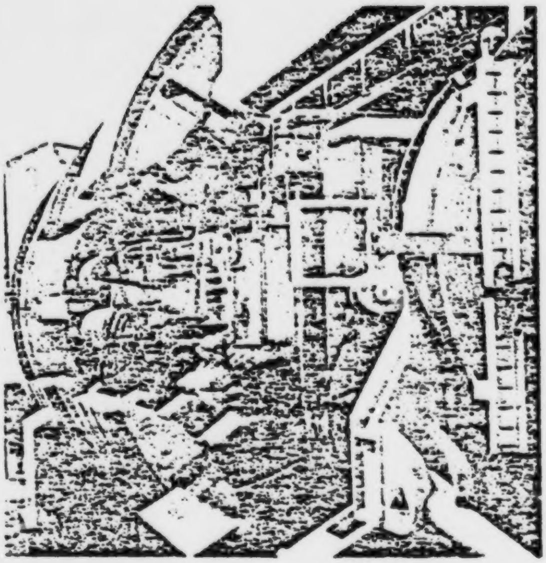


Fig. 6. Two sighting mechanisms were fitted on the machine.

Nine months to complete

It took some nine months to complete the inclined gallery, having taken into account the suspension of work during one month which is customary on Swiss mountain drilling sites, the month of November, 1968 (installations in the inclined gallery) and a 13m long faulty zone penetrated by conventional methods. The drilling rate averaged 5·30m per working day on a two-shift basis. The maximum daily advance was 15·75m, the maximum monthly rate achieved being 140m. The working hours may be split on the individual operations as follows:

1. drilling	50%
2. cutter maintenance	10%
3. installations	34%
4. repairs and breakdowns	3%
5. reaming	3%

The faulty zone mentioned was not taken into account since it has little influence on the overall advancement.

What is striking is the small portion of repairs and breakdowns of any kind. This figure comprises both disturbances of the installations in the gallery and the failures of the machine proper, the latter being practically nil. The operators of the machine took care of the maintenance during drilling. The machine operated perfectly well from the very first day and did not require any starting period. This came as a surprise for all concerned, all the more since the machine was a prototype that had never before been used for such kind of work.

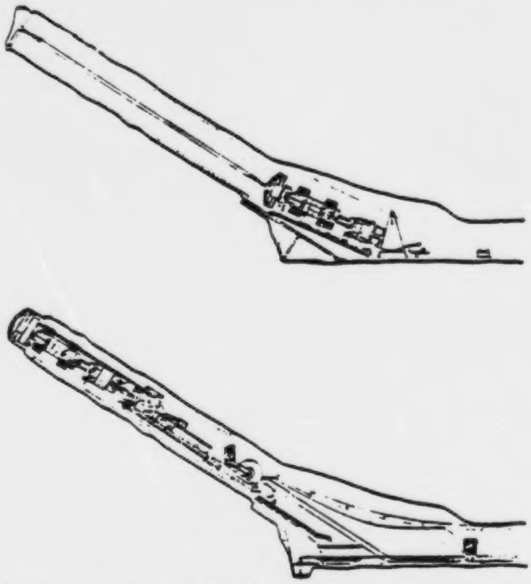


Fig. 7. An automatic locking device prevented the machine from sliding backwards.

Despite its weight of some 90 tons, the machine had no difficulty in operating in an inclined position. The automatic locking device designed for this purpose (Fig. 7) functioned at any time, which was particularly important for the operator and for the workmen on the floor, dispelling their initial fears that the machine might slide back. The working hours of item 2 making up some 10% of the total hours, were taken by cutter checks and exchanges. The cutter checks totalled 54, i.e. one every 21m. Taking as basis an average exchange of four cutters each time, this implies that a total of 216 cutters were removed and fitted. With a weight of 80kp each cutter, this means that a total of 17 tons were transported manually to the tunnel face and back again. That is by far too much. Thanks to the tests, however, that were made at the time the cutter life could be increased. Along with this went a drastic cut in the production cost of the cutters. The cutter costs

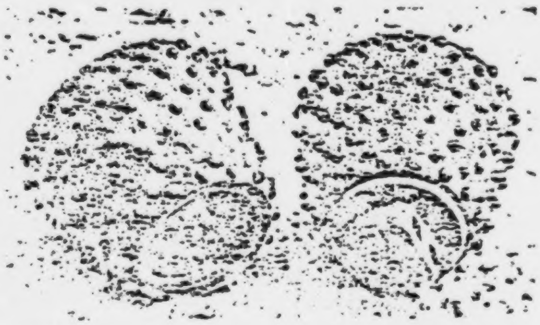


Fig. 8. Comparative wear on new and used cutters

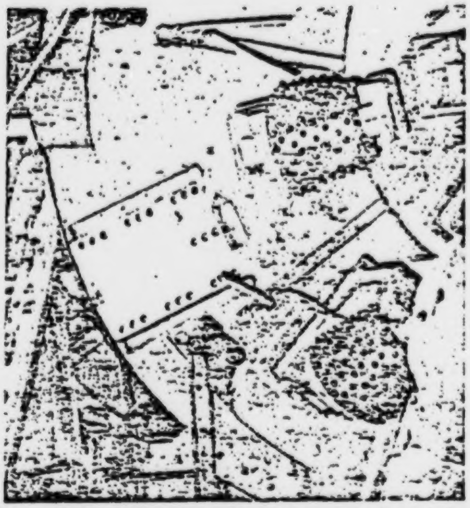
incurred in our application remained within the frame of the amount set aside for this end. They would have been some 20% lower if we had already had the cheaper cutters.

Maintenance of cutters

The cutter question remains the decisive criterion of drilling in hard rock. Careful maintenance of the cutters is indispensable. What matters in addition to the lubrication is to keep an eye on the fitting of new cutters on a head studded with slightly worn-off cutters. Wear of a few millimetres lessens the efficiency of the cutter, but it does not mean that the cutter no longer has any value. It is much worse if a new cutter is fitted in a row with used ones. With an advancement of 50cm/h and 9 revolutions/min. of the drill head, a penetration is obtained into the face of some 0·9mm per revolution. One can imagine the rapid wear a new cutter fitted with worn-off ones is subjected to (Fig. 3). All the more if you bear in mind that the cutters are fitted on the drill head so as to obtain the same penetration per revolution. This implies that only equal cutters should be fitted in one and the same row. A rearrangement of the cutters on the drill head pays off at any rate.

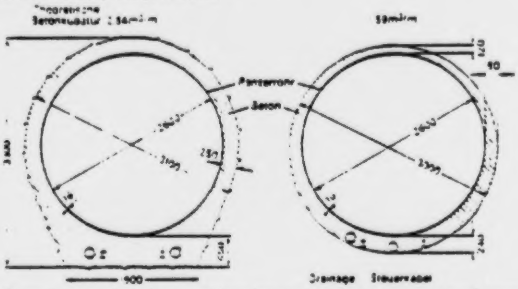
The item 'cutter maintenance' comprises among other things the time taken by twice redressing the saddles which accommodate the cutters. Having drilled some 400m, we discovered that the cutter fixing bolts were shearing off. It turned out that the saddle surfaces (Fig. 9) were hollowed out up to 2mm. The cutters normally fit these faces tightly to transmit the thrust. The shearing forces released thereby in the bolts could not be absorbed any longer. As a consequence we were forced to remove all the cutters and to reface the saddles with the help of special electrodes. The same happened after another 400m, necessitating again a refacing of the saddles. In order to prevent this troublesome disturbance, it will be necessary to have a special finish for the thrust faces of the cutter mounts in future. The main reason for the wear is, however, not to be found in the static stress, which is of only minor significance (some 150kg/cm²).

What is much more disagreeable are the dynamic forces, the so-called 'chatter', that is caused by clearance of the bearings. This is why one should not hesitate to drill with the maximum permissible thrust, i.e. approx. 8 tons permanent load on each cutter. This thrust can be absorbed by the bearings in any case, and the 'chatter' is reduced in proportion to the increase in thrust. However, there will still be required a host of similar projects to find out the exact ratio bit wear and thrust, i.e. drilling rate in order to calculate the optimum thrust. To sum up, the arrangement of equal cutters in the same rows is of decisive importance for the cutter life.



Above: Fig. 9. Saddles had to be refaced when cutter fixing bolts were found to be shearing off.

Below: Fig. 10. A concrete saving of 2·45m³/m was achieved from absence of overbreak and a reduction of theoretical concrete thickness due to the fault-free rock.



Item 1 contains the working hours by the operator and two workmen (who transported the cuttings to the portal and from there by lorry to a dump) as well as part of those by the winch operator. The installations (supply lines, rails, control of material flow, duct) required quite a lot of time. This is, however, not at all surprising in view of the restricted transport facilities in an inclined shaft. The figure does not include the time for the preparation and removal of the machine.

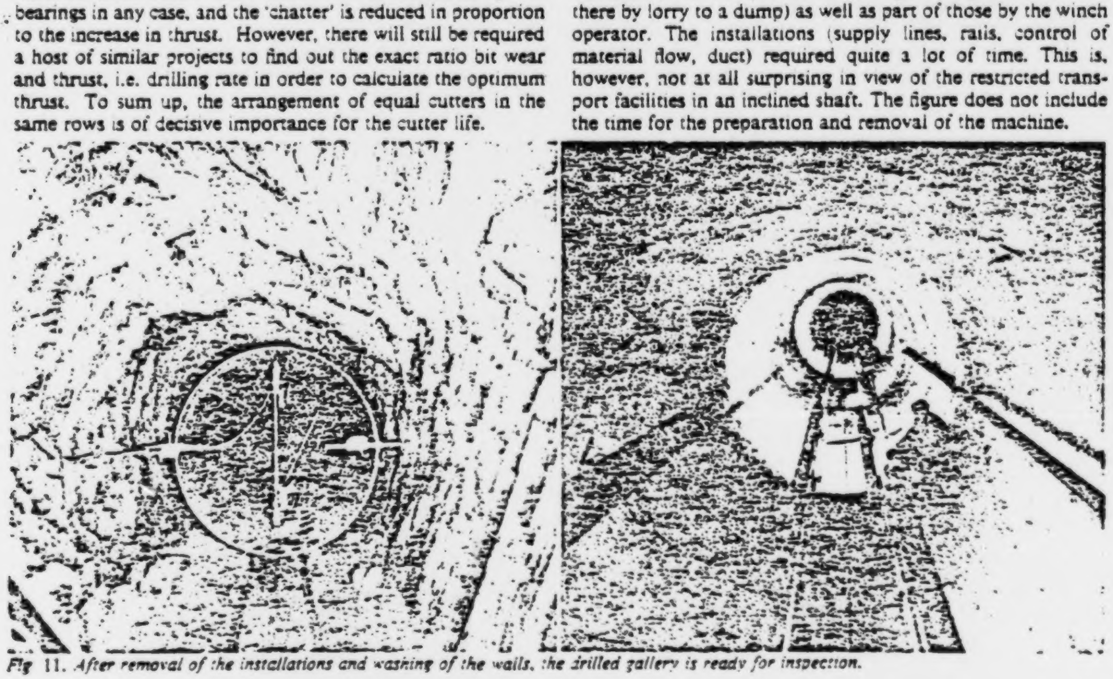


Fig 11. After removal of the installations and washing of the walls, the drilled gallery is ready for inspection.

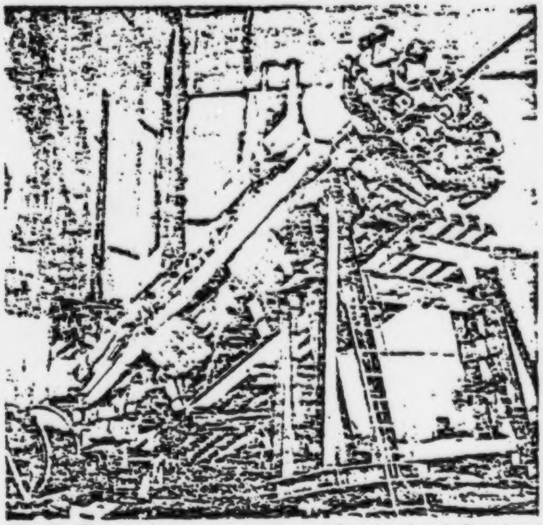


Fig. 12. The machine is winched into the chamber before being disassembled and removed.

Measuring efficiency

The efficiency factor, i.e. the ratio net drilling time to total working time is calculated as follows (the descent to the gallery heading as well as the lunch break of the workmen are included in the figures):

$$\lambda = \frac{T_1}{S \times t} = \frac{2100}{413 \times 10} = 0{\cdot}51$$

T_1 operating hours of machine
S shifts
t official working time per shift

Just for comparison, this factor, which is of paramount importance for economical drilling, is 0·6 in horizontal galleries. It may be increased by further improving the installation in the gallery, but above all by improving the cutters with the objective to reduce the controls to a minimum.

Some 1,108m of the 1,140m long gallery were drilled. The remaining 32m were driven by conventional methods to start the machine and to penetrate the faulty zone mentioned earlier. The advance speed averaged:

$$v_m = L/T_1 = 1108/2100 = 0{\cdot}53\text{m/hour}$$

The effective speeds vary between 37 and 142cm/h depending on the hardness of rock. They were achieved by a thrust on the drill head of 160 tons, that was kept constant and increased to 190 tons after the 'chatter' of the cutters was discovered. With equal distribution of thrust, the load on each cutter lay in the order of 6·15 and 7·30 tons respectively. Taking the uneven wear into account, the real load on certain cutters should have been much higher.

The power requirements for the drilling proper were in the order of 40 kWh/m^3, with a power intake of 130kW and an installed capacity of 300kW. It follows that the machine had a reserve of some 55%. Considerably higher was the power required for pumping some 5 l/sec. of drilling mud and some 3 l/sec. of mountain water from the heading into a natural settling basin and, last but not least, for the ventilation.

Concrete saving

Let me revert once again to the sizeable concrete saving that is one of the great advantages drilling has over conventional methods. In our case, we saved some 2·45m^3/m which resulted firstly from the absence of overbreak which is inevitable in blasting and secondly from the reduction of the theoretical concrete thickness on account of the absolutely fault-free rock (Fig. 10). Taking as basis a concrete price of 140sfr/m^3, and an average volume of excavation of 6·7m^3, the concrete cost saved totals 51sfr/m^3. Bearing in mind that explosives, fuses and drill rods are superfluous (which makes another 15sfr/m^3) one can see that the cutter cost is largely covered by these savings alone.

The small crew (only 5 to 6 men for each shift) meant a low wage fraction. This gains in importance in view of the fact that operations such as reshaping, rock cleaning, etc. are rendered superfluous. After removal of the installations and washing of the walls, the drilled gallery is ready for inspection (Fig. 11).

The undoubtedly low cost of the supports, too, plays an important part. A gallery passing through sound granite is not of course the best example to show the savings. However, it should be noted that we did not have to fit a single rock bolt or any other support though there were several fissures and spall-prone rock zones. Only the faulty zone of some 13m length (see *Tiefbau*, May, 1969) required some 20m^3 of gunned concrete. It may be taken that these fissures, let alone the spall-prone strata, could not have been bridged without supports if the gallery had been driven by conventional methods.

Dismantling the machine

Let me close with respectively the disassembly and the removal of the machine from the inclined gallery. The gallery ended in an underground winch chamber accessible through a 500m long adit. The machine moved onto a platform (33° incline) in the shaft extension partly of its own strength and partly with the help of a travelling block and a special support on the bottom required to make up for the absence of clamping possibilities in the section (Fig. 12). Machine and platform were then lowered hydraulically step by step till they had reached a horizontal position. The machine was turned by some 90°, loaded on two powerful wagons and drawn on rails to the portal. The trailers accommodating the downhill safety, the electrical equipment, the transformer, the cable drum and the silo for coarse cuttings were pulled out by means of a powerful travelling block. The entire dismantling, to start with the fitting of the platform and finish with the loading of machine and trailers on deep loaders, took as little as two weeks.

Now that the inclined gallery has been completed, we may justly say this undertaking turned out a success. All concerned are proud not only of having been right after all, but also of having helped improve the methods of tunnelling by their deed of pioneering.

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SHAFT SINKING

20-foot-diameter blind shaft drilling

R.P. CARONE
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ABSTRACT

Shaft sinking by blind shaft drilling offers the mining industry a different approach to shaft construction. Benefits of blind drilling include less construction time and fewer safety problems while being economically competitive with conventional sinking methods under certain conditions in soft water-bearing formations.

Technology and equipment are available today to drill blind shafts up to 20 feet in diameter and over 3,000 feet deep. This is an extension of proven techniques developed by private industry. To drill a 20-foot-diameter hole requires a specially manufactured hydraulic drill unit with a rated hoist of 2 million pounds and 500,000 ft/lb of torque.

This brief paper will discuss the following aspects of blind shaft drilling:

- *benefits;*
- *equipment for drilling;*
- *circulation methods;*
- *circulation rates;*
- *shaft alignment; and*
- *progress rates to be expected.*

R.P. Carone

Robert P. Carone, senior design engineer with Santa Fe Engineering and Construction Co., graduated from the University of Southern California with a B.Sc. in petroleum engineering in 1977. He is currently involved in the design of steel and concrete liners or a combination thereof to obtain a competent economical liner. In pursuit of these duties, experience has been gained in the use of the LINEVAL program, ANSYS and other computer modelling techniques. His background consists of offshore drilling experience, including cementing programs, lodging, casing design, formation breakdown tests, drill-stem testing and other associated drilling practices.

D.A. Whitley

David A. Whitley, project engineer with Santa Fe Engineering and Construction Co., graduated from Montana State University with a B.Sc. in civil engineering in 1972. His eight years of drilling engineering includes three years working in production and drilling for a major oil company in Louisiana, two years in charge of drilling and production engineering for a small independent oil company in Montana and two years as senior drilling engineer for an engineering contractor involved with the Atomic Energy program for the DOE. His experience includes designing casing strings, writing drilling programs and cost estimates, and monitoring rig operations drilling holes from 4 inches to 240 inches in diameter. His present responsibilities include the preparation and implementation of drilling, casing and cementing programs for large-diameter shafts, engineering for the installation of large-diameter shafts, including mine and oil storage shafts, offshore platform foundations and water wells, the development of new techniques for lining mine shafts and the monitoring of field operations, assisting in solving any problems which develop.

Introduction

Shaft sinking by blind shaft drilling offers the mining industry an improved alternative method for shaft construction. The drilling of 20-foot-diameter shafts to depths in excess of 3,000 feet can be done by utilizing equipment specifically designed and manufactured for this purpose. This equipment is a distant cousin, at best, to standard oil well drilling rigs. The actual configuration of these drilling machines will be discussed in a following section.

To most people, the drilling of mine shafts may seem to be a new or at least recent technology. However, as early as the 1920s, German and Dutch miners were drilling 24-foot-diameter shafts for coal mines. What led miners to resort to drilling were ground conditions that still plague conventional shaft sinkers today: soft, water-bearing formations. The major drawback to these early attempts was a low penetration rate due to poor bit cleaning. This, however, was not the end of early shaft drilling activity. From 1936 to 1950, a number of mine shafts were drilled by the calyx or coring method. This method employs a bit with a large bore down the center. As it drills, it cuts a core. When 5 to 10 feet have been drilled, the core is removed. The disadvantages of this method were the time and work involved in breaking the core free, and it was inadequate in soft water-bearing ground.

Today's shaft drilling technology has most of its roots at the United States Nuclear Test Grounds in Mercury, Nevada. It was here that the magic combination of a real need and virtually unlimited funds provided great strides in the science of shaft drilling. Because of the ban on atmospheric testing of nuclear

Keywords: Shaft sinking, Drilling, Shaft drilling, Blind shaft drilling, Rotary drilling.

devices, the now defunct Atomic Energy Commission made the decision to set the devices off at the bottom of large-diameter shafts that had been backfilled. When the program was initiated, the shafts were drilled using techniques most familiar to the contractors hired for the job; that is, oilfield technology. It soon became apparent that new systems were going to have to be designed to make shaft drilling feasible. From these first truly serious and costly attempts, shaft drilling has developed into an experienced and qualified technology.

Benefits of Blind Shaft Drilling

The most prominent advantage of shaft drilling over conventional shaft sinking is speed. In the New Mexico mineral belt, a 20-foot-diameter shaft 3,500 feet deep can be expected to take four years to complete. Drilling methods could cut this time down to one year. This one-year period includes lining operations. The reason for such drastic differences in time is that drilling incorporates penetration and muck removal into one continuous operation.

Safety is also significantly increased, as the workers do not enter the shaft until it is completely lined. This also adds to the speed of operations, because adverse down-hole conditions, such as heat, water and humidity, do not affect the workers' performance.

On an economic basis, shaft drilling is competitive with conventional shaft sinking, looking only at the bottom-line costs to construct a particular shaft under certain ground conditions. However, if the time value of money is considered, the scale, in these cases, leans heavily toward drilling methods. It is generally accepted that a dollar today is better than a dollar tomorrow. By employing drilling methods, such a shaft can become productive in one-quarter the time it would take to sink it conventionally. This greatly improves the cash flow picture of the proposed shaft construction.

Another advantage to shaft drilling is control of ground waters and unstable formations. These problems can seriously hinder conventional shaft sinkers and, in some cases, the costly procedures of ground freezing must be employed. When drilling, these problems are easily handled by manipulation of the drilling fluid. Such conditions are countered by weighting up the fluid. This increases the hydrostatic head, which exerts an opposing pressure on the aquifer or zone of instability and prevents inflow of water and/or debris.

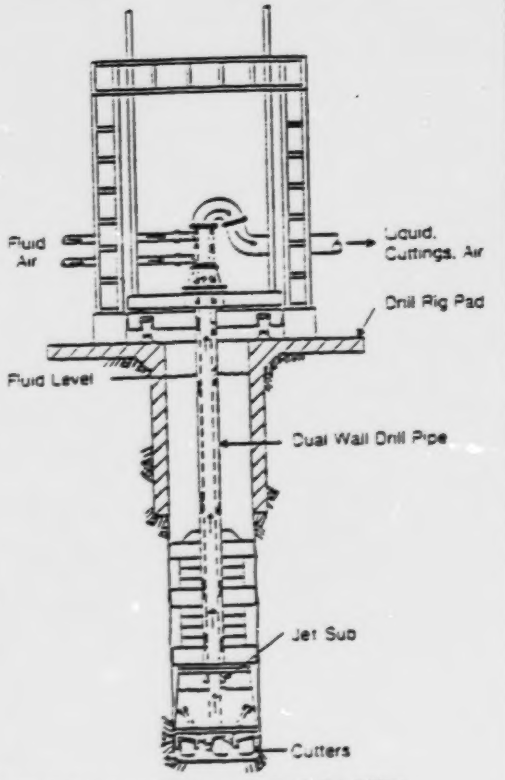


FIGURE 1. Air-assisted reverse circulation drilling.

Drilling Methods

The three factors that affect penetration rates are bit weight, rotational speed of the bit and bit cleaning. Of these, the real determining link is bit cleaning. This is due to the ratio of already broken material to fresh formation exposed to the cutters. The elements that affect how efficiently a bit can be cleaned are fluid turbulence under the bit, fluid velocity under the bit and viscosity of the drilling fluid. Various designs and devices have been implemented. Some of these are hydraulic jets, bit skirts, cutter pumps, baffles and recessed cutters, all of which have met with varying degrees of success throughout the industry.

Turbulence is necessary to help pick up broken particles from the drilling surface and, once lifted, keep them in suspension as they are swept toward the pickup aperture of the bit. Fluid velocity helps to create turbulence and currents toward the pickup. Fluid viscosity aids cleaning because the more viscous the fluid, the longer the suspension time of the cuttings.

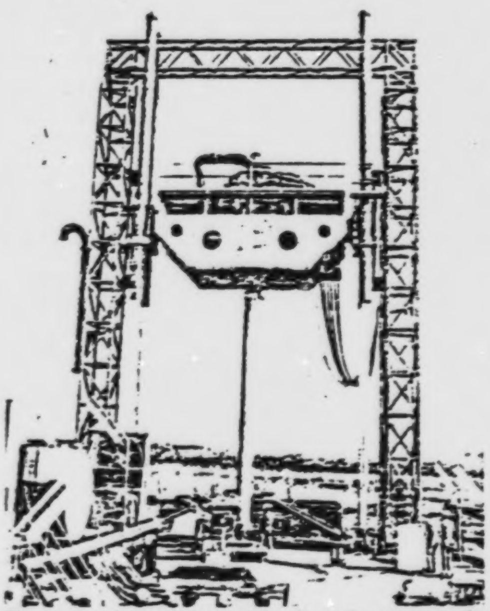
Fluid velocity and viscosity are also the elements necessary for efficient transport of the cuttings to the surface. Slip velocity must be considered when dealing with the transport of solid particles via a fluid in a vertical stream. Slip velocity describes how fast a particle of a given shape and density will fall through a fluid of a given density and viscosity. To convey cuttings up through the drill pipe via the drilling fluid (which in some instances could be air), the fluid velocity must exceed the slip velocity. The more viscous the fluid, the less the fluid velocity must be. Usually, the maximum particle load any system can carry is approximately five per cent volume. For a given size of hole, there is an optimum fluid velocity that must be maintained, because too much will cause erosion of the bore-hole wall. In large-diameter holes, this "washing" occurs around the bit, which can damage the bit itself.

In small-diameter holes, such as in oilwell drilling, direct circulation is used. Here, the drilling fluid is pumped down the drill string and returns up the annulus of the bore hole and the drill pipe. Because the cross-sectional area is small, the required fluid velocities are easily obtained. However, when drilling holes of large diameter, usually anything over 30 inches, the amount of energy required to maintain particle lift becomes enormous. Under these conditions, reverse circulation is used. As the term implies, the drilling fluid and cuttings are returned up the drill pipe. When drilling with direct circulation, mud pumps force the fluid down the drill pipe. When drilling with reverse circulation, an air system is used to lift the fluid up the drill pipe. This system was first developed in 1795 to produce water from wells and underground mines. It works under the same principle that causes champagne to flow from a newly opened bottle. Air-assist reverse circulation systems vary. Single-wall, dual-wall or triple-wall drill pipe can be used. The simplest system is the single wall. In this case, an air line, usually 3-inch pipe if 13⅜-inch drill pipe is used, is run down the drill string approximately 300 feet. Air is injected down this 3-inch pipe. As the air rises it expands, forcing fluid up with it. Drilling fluid is returned via the 13⅜-inch drill pipe. In the other two systems, air and drilling fluid are pumped down the drill pipe, either together or separately, and returned up the center. The advantage to these last two systems is that hydraulic energy can be directed on the cutting surface via jets.

Fast, efficient drilling of large-diameter shafts can now be accomplished with the CSD-300 shaft drill rig. The CSD-300 has the capability to blind drill shafts 20 feet in diameter in excess of 3,000 feet deep. This means faster mine openings at a competitive cost with conventional shaft sinking techniques.

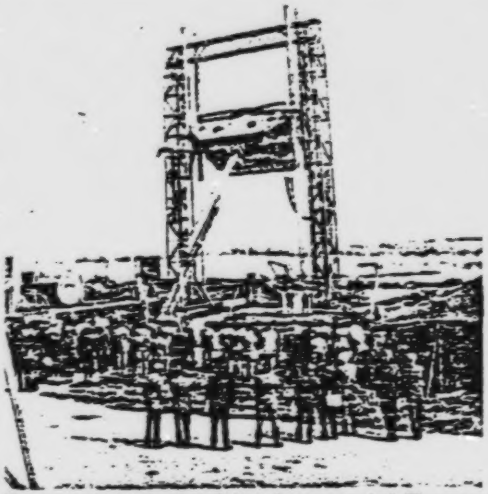
Work in Canada is currently being identified by Santa Fe Engineering & Construction Co. of Orange, California. Santa Fe is an international drilling and construction contractor with extensive experience in large-diameter drilling. Various methods of lining the drilled shaft have been developed which provide the operator with a dry shaft.

The CSD-300 rig was manufactured by Hughes Tool Company at their plant in Corsicana, Texas.

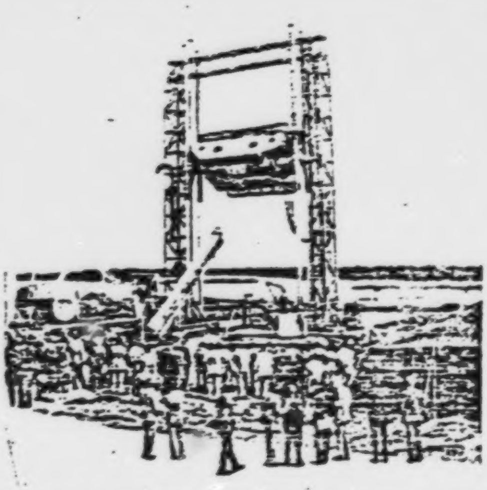


The power swivel (center) generates the cutting torque to the bit. Six hydraulic motors on the swivel provide 500,000 foot-pounds of torque. The power swivel is raised and lowered on hydraulic cylinders with 2,000,000 pounds hoist capacity.

This is a 12-foot-diameter bit complete with a dressing of mill tooth roller cutters. The 20-foot-diameter bit being manufactured will be assembled at the construction site in sections. Note the mud jet nozzles running up and down on this bit.



The new joint of pipe threads into the power swivel. The power swivel will then rotate back to vertical and thread the new pipe into the assembly hung-off below on the power slips.



The power swivel rotates in order to receive a new joint of 20-inch-O.D. pipe. Simultaneously, the pipe loader prepares to offer the joint of pipe.

Figure 1 illustrates the principle of air-assist reverse circulation.

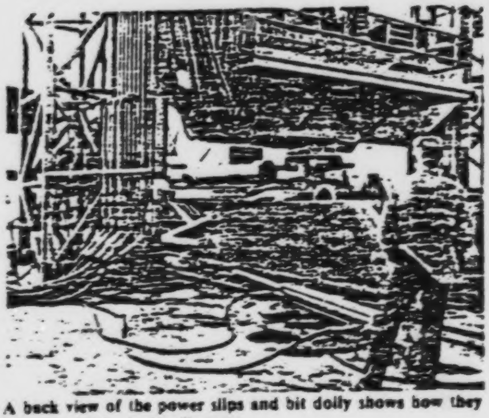
Bit configuration has probably undergone more evolution than any other aspect of big-hole technology. When modern-day shaft drilling began back at the Nevada Test Site, the conventional method of drilling large holes was to drill a hole of moderate size (30 inches) to total depth, then make a number of passes with a hole opener until the desired size was reached. However, if the target diameter was reasonably large, this process could take considerable time. To expedite operations, a bit was designed to perform the task in one pass. Essentially, this was a bit with the various-sized reaming attachments travelling behind it. This apparatus worked well. However, keeping the hole straight proved to be a problem. Apparently, each level of cutters would penetrate at a different rate, depending on the formation each was cutting at any given time.

Changes continued, with emphasis on a design that would facilitate producing a straight shaft. This resulted in a concave bit, with the gauge cutters (cutters on the periphery) approaching the formations first and the center cutters following. This indeed is the best design for straight holes, however penetration rates are unacceptable. Because of the upward flow of the drilling fluid across the cutting surface, gravity hinders the sweeping of the chips toward the pickup. This increases the number of times the cutters are exposed to chips rather than clean formation and, therefore, the penetration rate drops.

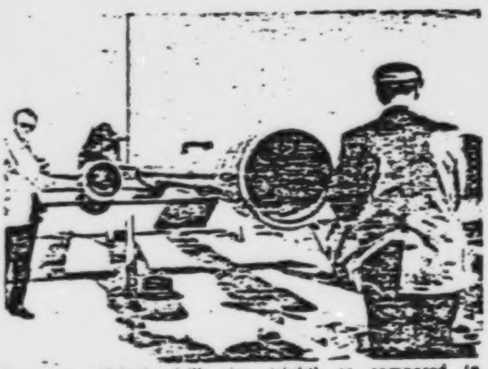
It is now generally accepted that a flat-faced bit is the optimum configuration for good cleaning and straight holes. However, there are still variations throughout the industry. One popular design in the European industry is to give the bit a slight cone shape with the center cutters set ahead of the gauge cutters.

Factors other than general shape have been considered and are still being investigated. As discussed earlier, these have to do with bit cleaning. Probably the most generally accepted system used to aid bit cleaning, and to some degree penetration, is the hydraulic jet. Hydraulic jets direct some of the energy of the mud pumps and air compressors against the cutting surface. The jets act to concentrate this energy at particular locations around the bit. This jetting action helps to pick up broken formation from the cutting surface, keeps the cutters from becoming clogged and, in some cases, aids in the penetration of soft formations. Large-hole bits vary extensively in placement of these jets. Some bit manufacturers prefer a "sweep-arm" design. In this configuration, all the jets are set in line radially from the center. Others prefer to place a jet behind each cutter. At present, no empirical data are available to determine the merits of either type of placement.

As mentioned before, turbulence and fluid velocity are needed for good bit cleaning. Different designs and devices have been experimented with to optimize these conditions. In some designs, the cutters have been recessed into the bit body to decrease the cross-sectional area between the cutting surface and the bit. This acts to increase fluid velocity across the bit, which increases turbulence. The drawback here is the bit body becoming damaged due to close proximity with the cuttings. Changing cutters is considerably more difficult in this configuration. Other methods have been tried to achieve the same end result, such as skirts and baffles. The benefit from these is still a matter of opinion. Cutter pumps have been used to help keep cutters free of debris. Cutter pumps are designed to employ the turbulence created by rotation of the cutter. Actually, they are no more than a housing for the cutters within which the turbulence is high due to the small areas enclosed by the housing. In general, use of such devices is a matter to be determined between the bit manufacturer and the client.



A back view of the power slips and bit dolly shows how they are actuated by hydraulic rams.



The new 20-inch drill pipe (right) as compared to $13^{3/8}$-inch-O.D. drill pipe. The drill pipe will be fitted with a $13^{3/8}$-inch inner pipe string for dual wall circulation. Each joint is 30 feet long.

Drilling Equipment

Originally, the drilling rigs used to drill large-diameter holes were converted oil-field rigs. These are adequate when dealing with holes around the 10-foot range, however they are seriously limited when attempting holes of larger diameter. To drill holes of 15 to 24 feet in diameter, a rig had to be designed based on a different set of principles. The rig had to be able to withstand the tremendous torques required of these immense bits. A system capable of delivering torques much greater than could be expected from the traditional rotary table had to be developed. Hoisting capacity had to be about double that of a standard oil-field rig. To meet these demands, a hydraulic rig was developed.

This rig (Fig. 2) stands much shorter than its oil-field counterpart, approximately 70 feet high. A hydraulic power swivel, capable of 500,000 ft-lb of torque, is used to rotate the drill string. This swivel threads into the box on the drill pipe and travels the length of the mast. This is the reason for the generally squatty appearance. When a new joint of pipe is added, the power swivel is high in the mast. The excessive torques encountered during drilling operations must then be restrained by the mast.

Hoisting is done by four hydraulic cylinders. These cylinders are 12 inches in diameter, with a 37-foot stroke. Each has a 500,000-lb capacity, yielding a rig capacity of 2,000,000 lb.

This rig is designed to use 20-inch-O.D. drill pipe. Each joint is 30 feet long and weighs more than 9,000 lb. To cope with such a cumbersome load, a special pipe handling system

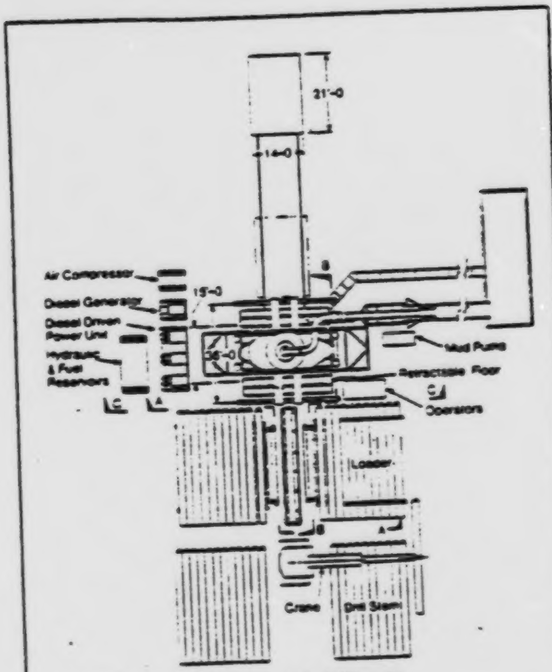

21'-0
14'-0
Air Compressor
15'-0
Diesel Generator
Diesel Driven Power Unit
Hydraulic & Fuel Reservoirs
36'-0
Mud Pumps
Retractable Floor
Operators
C
A
Loader
B
A
Crane
Drill Stem

FIGURE 2. Large-diameter shaft drilling rig.

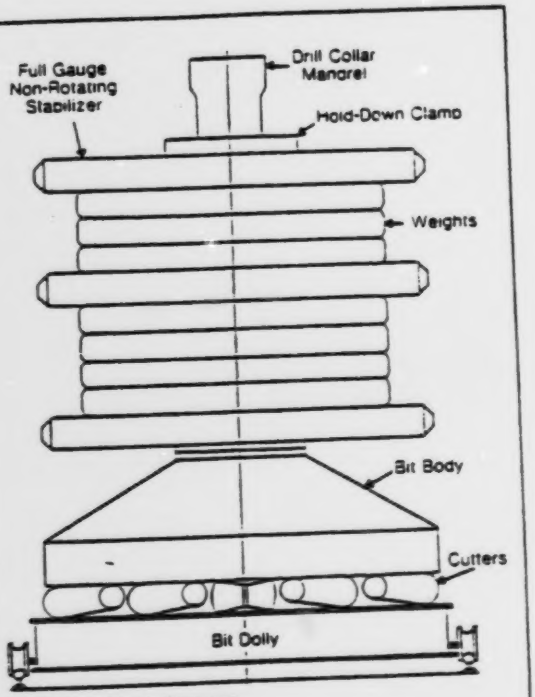

Full Gauge Non-Rotating Stabilizer
Drill Collar Mandrel
Hold-Down Clamp
Weights
Bit Body
Cutters
Bit Dolly

FIGURE 3. Shaft alignment.

was developed. No men are involved in this system other than to apply the thread lubrication.

Hydraulic power is supplied by six pumps powered by electric motors. These pumps are mounted on a single skid and are manifolded together. That is, they can be directed in any combination to any part of the rig, hoisting rams, power swivel and so on.

The downhole drilling assembly consists of a drilling mandrel, drill bit, stabilizers and weights (Fig. 3). The total weight of the drilling assembly can be 1,000,000 lb. The drilling mandrel is a heavy-wall 25-foot joint of pipe used to hold the various components together. The bit is attached to a "weight stool", which has a 132-inch flange that attaches to the bit. The flange is large to accommodate the required drilling torque. A non-rotating stabilizer fits around the weight stool.

Stabilizers are used to control deviation during drilling operations. There are two basic designs: rotating and non-rotating. Rotating stabilizers, as the name implies, are designed to rotate with the drilling assembly. Essentially, this is a body built slightly under gauge with cylindrical rollers set around the periphery. Non-rotating stabilizers are designed to remain stationary as the drilling assembly rotates within it. They offer better protection against hole deviation. This is because of the mechanical action on the bore-hole wall imparted by the rotating style. If a hard zone is encountered and penetration rates drop, this mechanical action may cause the hole to enlarge substantially. This could result in deviation. However, non-rotational stabilizers do not create this agitation, and chances of hole enlargement are diminished.

Ideally, stabilizers should be placed as close to the bit as possible and throughout the drilling assembly. This prevents any buckling of the mandrel, regardless of how slight, that could result in hole deviation.

Above the weight stool rests the weights. The type of weights used are referred to as "split donut weights." These weights are stacked throughout the mandrel. Each weight consists of two to six parts. The weights are split for ease of handling and transportation. Each part interlocks with the others for easy assembly. The total weight of the drilling assembly can be varied by adding or subtracting weights. These weights serve two purposes. First, they create the necessary weight for cutter penetration. Second, the extreme weight produces a "plumb-bob" effect to help keep the hole on course. The drill pipe is thus always in tension, never in compression.

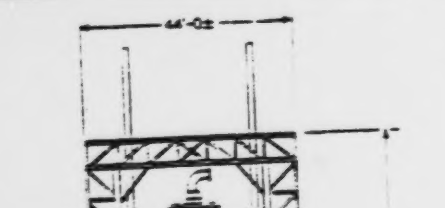

44'-0±
70'-0±
33'-0
7'-0
20'-0
Shaft

FIGURE 4. Cross section A-A, showing surface ancillary equipment.

Ancillary Equipment

A typical surface layout of the ancillary equipment is shown in Figure 4. This is meant for illustration only, as the layout for the equipment would have to be site-specific for each shaft. The ancillary equipment required for drilling a 20-foot-diameter shaft is described below:

Air Compressor:	Sufficient to deliver 3,000 ACFM at 1,000 psi discharge pressure.
Fluid Pumps:	Sufficient to deliver 4,000 GPM at 1,000 psi discharge pressure.
Power Generation:	Sufficient to generate 4.5 to 5.0 megawatts of AC electricity (or equal mechanical power).
Hydraulic Power Unit:	Six variable-volume pumps driven by three 350-horsepower AC electric motors.
Pipe Racks:	To handle 3,500 feet of 20-inch-O.D. drill pipe or drill rods.
Mud System:	(See Fig. 5) The mud system will handle 4,000 GPM of fluid and 2 tons of cuttings per minute. Components of the mud system are: o Air/mud separator to knock out the free air. o Gumbo separator to remove particles larger than ½ inch in diameter. o Shale shaker or vibrating screen to remove particles larger than 125 microns. o Desander and desilter, which are cone-shaped hydrocyclones used to remove particles to 20 microns. o Conveyor equipment, which could be used to move the cuttings (muck) off of the shaft site, if required.

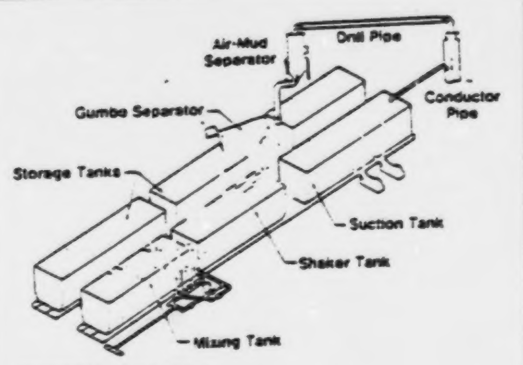


FIGURE 5. Mud system.

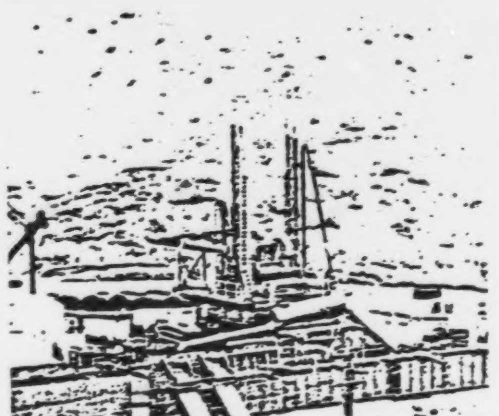


FIGURE 6. Hughes Tool Co. CSD-420, capable of 8-ft-diameter shafts.

Expected Progress

As mentioned earlier, one of the advantages of drilling shafts is the speed with which the shaft can be constructed. The time required can be one-third to one-fourth the time required for conventional sinking, especially in poor ground conditions. The progress rates for a 20-foot-diameter hole can be 1 to 2 feet per hour. The average rate, as predicted by Fenix & Scisson, Inc. in a study prepared for the United States Department of Energy and completed in January 1980, was 1.24 feet per hour. This is based on large-diameter hole drilling data

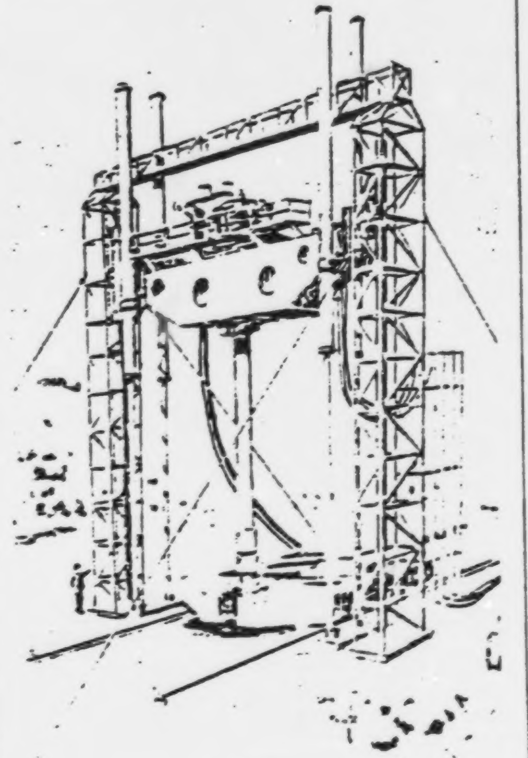


FIGURE 7. Hughes Tool Co. CSD-300, capable of 20-ft-diameter shafts.

compiled from 1963 to the present. The most recent shaft drilling experience is the construction of a 2,145-foot-deep shaft for the Crown Point, New Mexico project. This shaft was drilled in 129 days, including 9 days of down time for equipment repairs. The shaft was in an area with high-head aquifers. Some of the aquifers are artesian. The time and cost for construction of conventionally sunk shafts in the area are approximately 75 per cent higher and 400 per cent longer.

Summary

Shaft construction by rotary drilling methods is a proven technology, with several thousand feet of drilled shafts haveing been constructed. The early methods were slow and expensive. With the improvements and developments of the last two decades, the drilling of shafts has proven to be fast and relatively inexpensive. The equipment is available to drill shafts up to 20 feet in diameter to depths in excess of 3,000 feet. The hydraulic equipment available today (Figs. 6 and 7) can be utilized to construct shafts faster, safer and less affected by underground problems than conventional shaft sinking techniques. The mining operator can realize economic benefit by utilizing shaft drilling as the method for shaft construction under certain conditions.

New book examines polymeric materials used in mines, and their fire safety

Fire Safety Aspects of Polymeric Materials: Volume 10, *Mines and Bunkers*, is a new book which was researched and written by a special committee of the U.S. National Materials Advisory Board and sponsored by the National Academy of Sciences.

The 185-page book examines the many uses and the fire behaviour of the polymeric materials being used in mine supplies and equipment. All types of synthetic and natural polymeric materials are discussed, including plastics, elastomers, adhesives, coatings, fibers and wood. These materials are evaluated with reference to flammability, smoke and toxicity.

Additional chapters cover fire dynamics and scenarios (what actually happens in mine and bunker fires and why), design criteria and hazard control in mines, and recommendations for improved fire safety.

This new volume will provide valuable information to designers and manufacturers of mine supplies and equipment, and suppliers of materials for these products. It will also be helpful to mine managers and safety, regulatory and fire-fighting personnel.

Fire Safety Aspects of Polymeric Materials: Volume 10, *Mines and Bunkers* (6x9, hardcover), is available for $25.00 per copy from the publishers, Technomic Publishing Company, Inc., 265 Post Rd. West, Westport CT 06880, USA.

Vancouver to host world chemical congress

The 28th Congress of the International Union of Pure and Applied Chemistry (I.U.P.A.C.) will be held in Vancouver, **August 16-22, 1981**. The president of the Congress is Dr. Gerhard Herzberg, Nobel prize-winner in chemistry of the National Research Council, Ottawa.

Among some 2,500 Canadian and international chemists and scientists attending will be many of world-wide reputation, including Prof. Manfred Eigen, Nobel prize-winner in chemistry of the Max Planck Institute, Goettingen, West Germany.

According to Glynn Michael, executive director of The Chemical Institute of Canada, one of the three sponsors of the I.U.P.A.C. Congress: "The main subject areas of this Congress will excite wide interest because of their topicality in this period of diminishing supplies of energy." The subject areas are:

- Chemistry in the Solution of the Energy Problem;
- Chemistry in the Study of Environmental Problems;
- Chemistry in the Extraction and Beneficiation of Resources.

Some 40 Canadian chemists from universities, industry and research centres have promised to deliver papers, as well as internationally known scientists from the U.S., West Germany, Great Britain, Switzerland, France and the Soviet Union, among others. Dr. Louis Renzoni, consultant to Inco Limited, is chairman of the Canadian National Committee for I.U.P.A.C.

I.U.P.A.C. is a non-profit association of national organizations, each of which represents the chemists of a member country. Its membership comprises 44 countries—Canada is represented by the National Research Council.

The most recent I.U.P.A.C. Congress took place in Helsinki, Finland in 1979.

University of Toronto holds 7th annual ore deposits workshop

This year's Ore Deposits Workshop will be held from the **14th to the 18th of December**. The program will feature:

Mississippi Valley Deposits, F.W. Beales and G.M. Anderson;
Nickel Sulphide and Platinum Ores, A.J. Naldrett;
Gold Deposits, D.A. Pretorius;
Massive Cu-Zn-Pb Sulphide Deposits, S.D. Scott;
Mo-W-Sn-Ta Deposits, E.T.C. Spooner;
Asbestos Deposits, F.J. Wicks;
Potash and Phosphate Deposits, E.T.C. Spooner; and
Chrome Deposits, I.H. Campbell.

The fee is $600. This workshop, however, is fully booked and it is unlikely that many places will become available. The waiting list is open for the December 1982 workshop and early reservations are recommended.

For further information, write to: Professor E.T.C. Spooner, Department of Geology, University of Toronto, Toronto, Ontario, M5S 1A1.

By COLIN F. SMITH

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Blind Hole Raise Boring

These remarks are about blind hole raise boring, as opposed to blind shaft boring or tunneling.

Tunneling by boring is not a new concept. A shield tunneling method for soft ground was patented by Brunell in 1818, and a steam driven mechanical miner was used in granite in Massachusetts as early as 1852. Col. Beaumount's famous tunnel borer was cutting a 7-ft diam tunnel in chalk under the English Channel, where a two man crew was advancing almost 40 ft per 17 hours, in 1884. However, most of the developments that have led to economic use of the concept have taken place during the past 20 years. Today, there are a dozen or so major manufacturers of tunnel and other specialized rock boring machines, supported by a substantial industry supplying the vital wearing parts, such as cutters and bits.

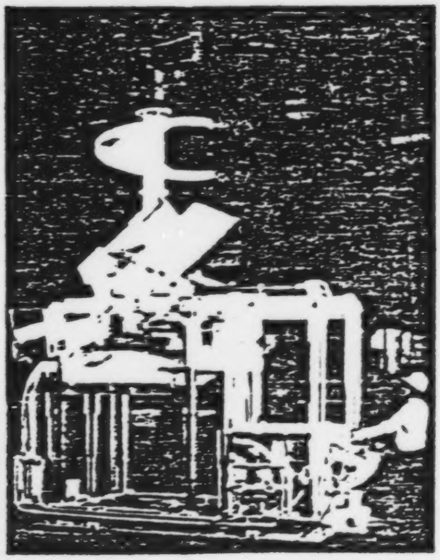
1950s saw start of boring steep openings

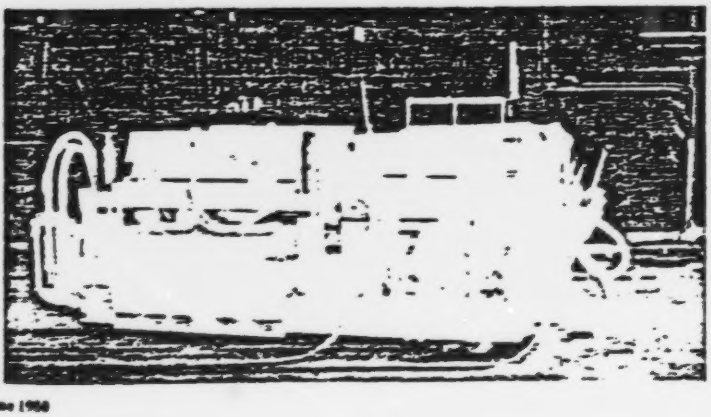
The boring of steeply inclined openings had its beginnings in the 1950s, and the precursor of the raise borer as we know it today came on the scene in 1962 in the form of the Robbins 31 R. The most recent technique introduced in the field of mechanical rock boring is that of boring blind holes from the bottom up. In South Africa this is known as "boxhole" boring. It employs a boring machine to cut a steeply inclined raise in a single full face operation which, unlike the better known raise boring technique, does not require a predrilled pilot hole. Calweld's VTB (vertical thrust borer), developed about 1967 for use in the uranium fields of New Mexico, was followed in 1974 by their Model BH 80, a boxhole borer which was designed for use in South African gold mines. Another recent development, the blind shaft borer, which must be distinguished from equipment which simply drills a large hole, is still in the development stage and has not yet been used commercially.

Torque, thrust transmitted by drill pipe in some units

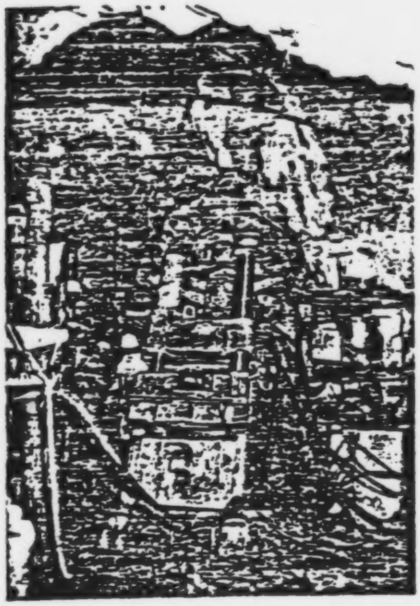
The Calweld and Subterranean boxhole machines both work on the principle of providing rotational and thrust forces from a machine location in the mine drift below the raise. Both manufacturers use threaded rotating drill pipe, through which both torque and thrust are transmitted.

Calweld's VTB, which was the first in the field, is still being made and used with very few changes in the soft sandstone mining application for which it was developed. The VTB is a self contained unit in which a rail mounted frame carries all the essential features of any blind hole raise boring machine. These are the power pack and controls, hydraulic cylinders to provide thrust, guide posts to align the thrust, hydraulic motors for rotation, a device for holding the drill string during addition of pipe and stabilizers, mechanical equipment for handling rods, stabilizers, etc., and a means for collecting, deflecting and loading cuttings.

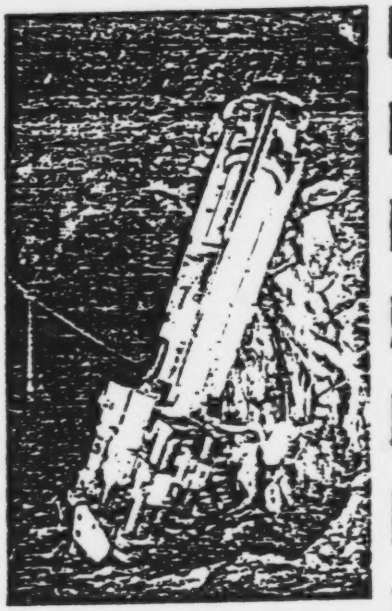




Both the Calweld and Subterranean boxhole machines work on the principle of providing rotational and thrust forces from a machine location in the mine drift below the raise. Pictured is Calweld's VTB collapsed for transport (left) and in its boring configuration (above)



Subterranean UR-60 boring an inclined hole



Robbins 52-R electric drive blind hole borer on site at Stilfontein gold mine, South Africa

First boxhole borer was experimental

The requirement of South African gold mines for inclined holes demanded a modification of the original concept. The first boxhole borer, the Calweld BH-80, was never more than an experimental machine, but many valuable lessons were learned from its operation on mines in South Africa's Orange Free State. The main specification changes made in the light of this experience were in installed power, reduced from 250 to 200 hp, cutter head rotation reduced from a maximum of 35 to 14 rpm, drill rod length reduced from 4 to 2 ft and drill rod connection changed from flangebolted to threaded connections.

The first Subterranean UR-60 was bought by Anglo-American Corp. in 1975, also for use in the Orange Free State mines. This unit was more compact than the BH-80 and introduced several new concepts to boxhole boring, notably the projecting pilot bit, or stinger, on the cutter-head which greatly facilitated collaring and directional accuracy.

The Robbins 52-R, introduced in 1975, differed from its predecessors in using a non-rotating, pin-connected drill pipe through which only the thrust was transmitted. Until recently, this system employed a hydraulic motor, attached to a gear case just below the cutting head, to provide rotation, the hydraulic fluid for driving the motor being pumped through hoses located within the drill string. The latest model, the 55-R, uses an electric drive for rotation, the motor being located in the same position, attached to the gear case at the rotating head.

All systems use non-rotating stabilizers

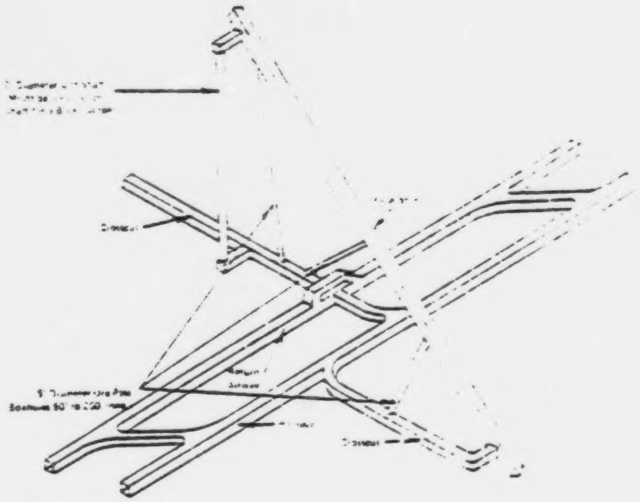
Stabilization/centralization of the drill string is of vital importance in boxhole boring. The flanged pipe connections and rotating stabilizer used with the earlier Calweld machines gave rise to numerous failures both of stabilizers and drill string, and rotating stabilizers tried with earlier holes drilled by the Subterranean machines were equally unsuccessful.

All currently operating systems utilize non-rotating stabilizers which take the form of arms fixed to the drill string through roller bearings in the case of rotating strings, and light continuous flights bolted to the non-rotating string in the case of the Robbins system. Rotating centralizers are used with the Subterranean system to provide added rigidity and gauge maintenance close behind the cutter-head.

While the two basic systems have obvious differences, they both offer advantages as well as disadvantages. The rotating drillstring concept has demonstrated the ability to bore a straight hole whereas the tendency in earlier models of the non-rotating system was for the bore to drift off line as the hole progressed. This problem appears to have been reduced somewhat by the addition of a front stabilization member, or stinger, to the cutter head. The non-rotating pipe sections and stabilizers are less expensive than the rotating pipes, which are necessarily made of high strength steel with precision-machined threads and shoulders to be able to transmit the combination of high torque and thrust loads. Stabilization of the rotating drill string is also more expensive.

Possibly the most important consideration in comparing the two basic systems is that of having the rotation gear box and drive up near the cutting-head and face. There is always the possibility of losing the cutter-head and some drill pipe in a raise, especially when working in blocky or unconsolidated ground. The risk of much more expensive loss is increased when the rotational drive unit is attached to the head. The loss of power to the rotation drive system, as hydraulic oil had to be pumped further up the hoses with increasing length of the bore, has been overcome with the introduction of electric drive rotation for the Robbins machine. Heat dissipation from the motor at the top of a blind hole does not appear to be a serious problem.

Typical mine stope development layout showing the main applications of boxhole boring in South African gold mines



A great deal of attention is still being given to optimizing stabilization of the drill string in boxhole boring. However, most of the causes of the failures which were experienced during the earlier development of the technique have been overcome by establishing the correct spacing between stabilizers and by achieving a better balance of tolerances and clearances within the stabilizer assemblies and between the stabilizers and the walls of the hole.

Practical hole lengths are 50 to 300 ft

In general, boxhole bored hole diameters range from 4 to 7 ft. The majority in South Africa are 5 ft diam but the new BH-60-SP machine is boring 7-ft diam holes. Hole lengths range from 50 ft, below which transport and set-up times make mechanical boring uneconomic, up to 300 ft, beyond which it has proved impractical to take boxhole boring. The majority of holes fall within the range of 100 to 200 ft long.

The time consumed in rod or stabilizer changing, both in the advancing hole and in withdrawing the drill string on completion of the hole, is a delay common to all machines. The efficiency of rod handling mechanisms is a significant competitive factor between different manufacturers, and operator skill is a major factor in minimizing this delay.

Working in sandstone of less than 5000 psi compressive strength, the vertical thrust borer can achieve an average of 4 to 5 ft per hour. Boxhole borers can achieve 3 to 4 ft per hour in rock of 35,000 psi, but rates can reduce to less than 1 ft per hour when rock hardness increases to 50,000 psi.

The best performances on record to date are approximately 44 ft per shift, 160 ft per week, and 400 ft per month, which includes one machine move.

Each unit produced by each manufacturer is an improvement on its predecessors, but at this point in time one can talk in terms of transport dimensions of the order of 9 ft long by 5½ ft above rail by 4 ft wide. Total machine weight, in the case of the latest UR-60, is less than 10 tons and the machine will operate in station heights of 10 ft for a vertical hole or 12½ ft for a 60° inclined hole.

Newer machines essentially self-erecting

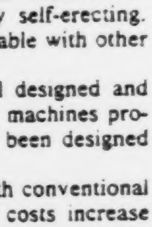
The evolution of the boxhole borer over the past eight years has brought it to the point where it is a useful and accepted mining tool. The overall size and weight of the units have been reduced so that they are acceptably mobile within the South African gold mines. Most of the newer machines are almost completely self-erecting. Reliability is approaching levels comparable with other underground mining equipment.

The original boxhole borers were all designed and built in the United States, but the later machines produced by all three manufacturers have been designed and built in South Africa.

Boxhole boring does not compete with conventional raiseboring for longer or larger holes; costs increase rapidly at lengths in excess of 200 ft and the longest holes bored to date are little more than 300 ft long. Hole diameters have not exceeded 7 ft, the average hole being between 4 and 5 ft in diameter.

The boxhole borer has proved to be a practical tool which has reached the stage of being used as a routine production unit by mining companies as well as being accepted as an economic unit of contractors' equipment.

While this new method of excavating steep raises is still in its infancy, much progress has been made. It is not the best method for all requirements, but boxhole boring is replacing both drill-and-blast methods and raise-boring techniques in an increasing number of mining applications. ◆



Colin Smith spent the first four years of his professional career, beginning in 1958, in mine production work and subsequently was engaged in shaft sinking and mine development in the gold mines of South Africa with Anglo American Corp. Since 1965 he has been involved in mining contracting, and for the past year he has been establishing a contracting organization in the United States for Roberts Union Corp. and Rockbor International Corp., of which he is vice president and general manager.

TechnicalPapersTechnicalPapers

Shaft Sinking Today— A Boring Business Tomorrow

Maurice Grieves

The great majority of shafts constructed today are still excavated by drilling and blasting, a method which changed very little in over 100 years until the introduction of the mechanical lashing unit and cactus grab by the South Africans, which enabled muck to be removed as fast as massive hoisting systems could handle it and resulted in very rapid rates of sinking. Record breaking month's performances were achieved at—Hartebeestfontein No. 4 shaft, October 1960—337.1 m; Western Reefs No. 4 shaft, October 1961—340.7 m; and Buffelsfontein eastern twin shaft, March 1962—381.2m. The method was very labor-intensive, requiring a crew of over 60 workers at the shaft bottom during the drill cycle. Safety precautions were strict, but in the drive to achieve rapid advance, cases of personal injury were still somewhat high because of the large number of people engaged in this potentially hostile environment.

The South African method, as it came to be referred to throughout the rest of the world, was adopted in the United Kingdom in the late 1950s in a modified form with greatly reduced manpower and nonsimultaneous sinking and lining, which was insisted on by the British Mines Inspectorate. In that instance, it was successfully used to sink the 7.3-m-diam concrete-lined shafts at Kellingley to 770 m depth, with rates of advance of over 90 m/month achieved, a British record at that time.

During the sinking of the 1.15-km-deep twin shafts at Boulby potash mine in the UK in 1970, the method was again used, but for the first time ever in Britain exemptions from the mining code permitted the use of crash beams, crash doors, jack catches, and semi-simultaneous sinking and lining techniques. New British shaft sinking records of over 120 m/month were achieved in both shafts.

Similar equipment and techniques were used in the early 1970s to sink several deep shafts in Canada, notably Creighton #7 and the Con zinc mine at Yellowknife in the NW Territories.

Today, this equipment is standard for deep shafts in the US and the rest of the world. However, with the tremendous escalation in mining labor costs, the impact of health and safety legislation, and environmental regulations, coupled with a very real shortage of miners willing to work in this exposed situation, it was apparent that an alternative to the labor-intensive conventional method of shaft construction had to be found.

Recognizing the trend is inevitable, one or two major German shaft sinking contracting firms began to take a fresh look at full face boring techniques applied to tunnels and raise bored shafts. The results were most encouraging. Tunnel drivage techniques using moles had developed considerably from Colonel Beaumont's original channel tunnel machine circa 1880 to the superbly engineered Priestley machine selected to cut the British side in 1975 and the double shielded Robbins Grandori borer on the French side of the English Channel.

Full face tunnel machines were being successfully used to drive uphill in inclined shafts in Austria and Switzerland. At Mapprag in Switzerland, the Demag mole drove the first (intentional) vertical transition and curve, and then went on to successfully complete the 730-m-long penstock shaft at an inclination of 35°. In Austria, the Wirth mole drove the Kaprun Glacier ski-lift railcar tunnel at record breaking rates of 457 m/MONTH (best 30 m/d) through green schist at an inclination of 29° for a distance of 3.35 km while the Hydro tunnel at Sarrelli in Switzerland was being driven by the Robbin's mole at an inclination of 35°. Simultaneously, extremely promising results were obtained using large assemblies of cutter discs on raise borer heads, such as the 4.87-m-diam × 460-m-deep shaft raised by Teton for Jim Walter Resources Inc.

Bearing in mind that most mine shafts in the future (unless in exceptionally competent rock) will require some form of lining, and the trend will be toward deeper shafts as the more easily accessible mineral deposits become exhausted, it was seen that normal raise boring had definite limitations in vertical accuracy, in the limitation imposed by the drill string on the available torque that could be applied to the cutter head, and in the risk of collapse of the unsupported shaft rock wall in friable or jointed and fissured ground, since it is not possible to apply any form of temporary support until the permanent lining is being installed. A further problem was the economics of installing a subsequent lining, necessitating setting up a headgear and hoisting arrangement approaching in size that required for conventional drill and blast sinking and lining.

Because of the economics, German contractors opted for a phased transition from drill and blast to the full face, rodless, out of the solid shaft mole, by starting off with a down the hole shaft boring machine—without a drillstring—but using a pilot hole to get rid of the muck.

Maurice Grieves is president of Thyssen Mining Construction Inc., Lakewood, CO. SME preprint 79-172, SME-AIME Annual Meeting, New Orleans, LA, Feb. 1979. Manuscript Jan. 1, 1979. Discussion of this paper must be submitted, in duplicate, prior to Feb. 29, 1982.

Table 1—Shafts Bored with V Mole 1 (5-m-diam)

Mine	Firm	Year	Depth m	Adv. m/d	Time (days)
Emil Mayrisch	EBV	1971	231	1.95	119
Walsum	RAG	1972	243	5.75	42
Zollverein	RAG	1973	227	9.00	25
Sterkrade	RAG	1974	196	13.08	15
Emil Mayrisch	EBV	74-75	228	8.00	29
Min. Stein	RAG	1975	202	9.60	21
Emil Mayrisch	EBV	1976	228	6.14	37
Rossenray	RAG	75-77	224	10.8	21
Emil Mayrisch	EBV	1977	228	7.90	29
Overall					
Incl. Emil Mayrisch		1971	2107	5.07	347
Excl. Emil Mayrisch		1971	1978	8.22	228

And so the concept of V Mole 1 was born. The reasons for this choice were fairly obvious:

- The mole, in principle, already existed. It was only necessary to transpose it from operating at an incline of 35° to operating standing on its head.
- Mining had been carried out in the Ruhr coal field for more than a century and mine workings were developed so extensively that wherever new shafts were required either for ventilation or mineral or man hoisting, access could be made available at the bottom of the selected shaft site. Thus, putting down a pilot hole caused no problems.
- Minor pilot hole deviations, which are bound to occur, raised no problems. As long as the cut muck could roll by gravity into the pilot hole the borehole could wander around a target area with a 1-m radius from the true vertical shaft center while allowing the mole to excavate a truly vertical shaft.
- By using a pilot hole to get rid of the muck, the single biggest constraint on the advance rate of the mole was eliminated.
- Access to the excavated rock is available at all times in a fully controlled situation, allowing immediate setting of temporary supports to fully comply with mining regulations. The supports considered were again traditional - steel rings backed by weld mesh as used to support horizontal full face mole drivages in the mines. The procedure was very simple, with no radical new concepts, thus insuring easy acceptance by mine owners and the mines Inspectorate.

In collaboration with Wirth Inc., the German tunnel machine and raise bore manufacturer, the first generation rodless shaft borer covering the range 4.5-5 m diam was built, and in 1971 the first shaft was drilled with the prototype V Mole 1. The machine was so successful that it was fully booked for a run of eight years. It was possible to plan machine moves so that prior to arrival at the next designated site, all preliminary work at the shaft top and bottom was completed, with the pilot hole already finished, ready for the V Mole.

Between 1971 and 1977, the machine drilled nine shafts between 200 and 300 m deep without a technical hitch at rates of advance averaging as much as 13.5 m/d. As an example, Ruhrkohle's Sterkrade shaft (200 m deep) was drilled out in just over two weeks. Table 1 shows the locations, depths, and rates achieved.

In constructing the shafts, steel supports were systematically set from a work deck, which, while attached to the V mole, was made independent of its rotational and vertical movement by a synchronized scissors elevating mechanism. Each support consisted of a 5-piece steel ring, with all joints of socket and bayonet type except one, which is a friction clamp. It is used to jack out the ring after the interlocking mesh panels have been placed behind it. By using a hydraulic jack with preset thrust it was possible to immediately prestress the support system, thus offering positive resistance to ground movement from the moment of setting. In every shaft sunk to date, this support method has proved totally adequate and has been extremely effective in preserving the structural integrity of the shaft in cases where the coal has been mined out below it at a later date.

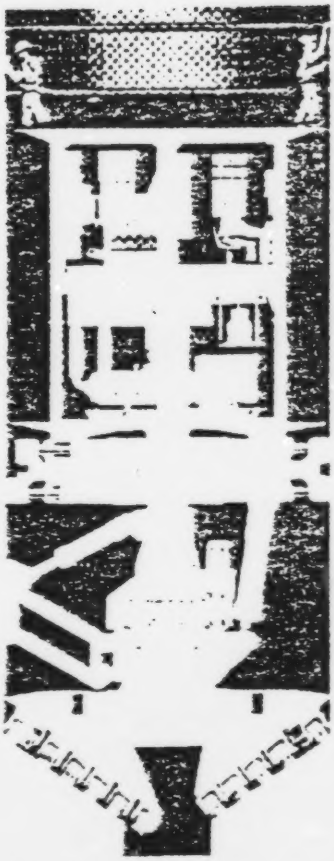


Fig. 1—The first generation V Mole 1 shaft boring machine.

In 1977, V Mole Mk 1 was withdrawn from active service, and replaced by Mole Mk 2, capable of boring 5-6.5 m diam. With it came improved efficiencies resulting from the elimination of most problems and defects evident in the prototype.

It was believed at the design stage of the first machine that considerable benefits would be derived by varying the rotation rate of the cutter head. In practice this was found to make very little difference. In V Mole 2 the temperamental variable speed hydraulic drive was replaced by a constant speed geared drive direct from electric motors at the optimum rotation speed determined by operational experience from V Mole 1.

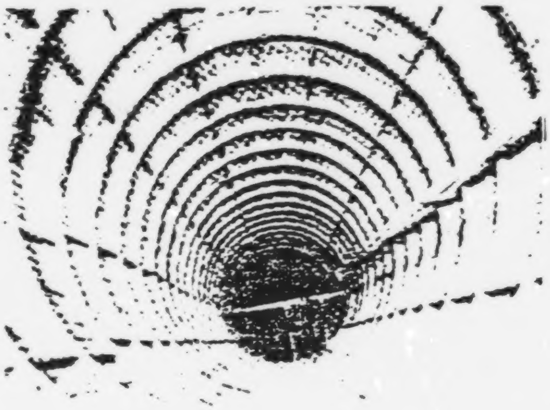


Fig. 2—Steel rings and mesh panels as shaft lining.

V Mole 2 was introduced at Preussag's Ibbenbueren mine in 1977 and drilled the 467-m-deep shaft in 44 days at an average rate of 10.6 m/d. At Ibbenbueren it was known that a 30-m-thick, highly pressurized gaseous bed of sandstone, liable to rock bursts, had to be bored through.

By modifying the laser control grid target system display used on V Mole 1, a remote controlled operational steering system was set up 300 m away from the top of the shaft. When the machine approached the hazard zone, it was operated completely by remote control. No one was on the machine during the cutting and advance operation. Gripper forces and cutter thrust were monitored by instruments in the remote location, and by varying these according to information portrayed on the sensor unit displaying the vertical position, the machine was taken through the 30 m hazard zone without anyone present at the shaft bottom.

The foresight in taking such precautions probably prevented the occurrence of a serious incident since seven rockbursts were triggered off releasing some 50 m^3 of sandstone that exploded as high pressure methane gas was released from the severely stressed pores of the rock. The machine suffered no damage, and no one was injured. Including the change to remote control, the complete shaft was finished at an average rate of 10 m/d.

Table 2 shows shafts constructed by V Mole 2. Victoria is a coal bunker (silo) and is lined with glass fiber reinforced concrete panels, into which the peripheral spiral is formed as a precast element. Erin was the first shaft to be lined using the same GRC panels set behind the machine as boring progresses. The operation of this machine was so successful that a third V Mole was ordered from Wirth. It was delivered in April 1980 and began work on a series of shafts in the Saar Region of West Germany. V Mole 3 will excavate at up to 8 m diam, thus allowing ample space for a 7.3-m-diam concrete lining, which is the standard at most major new mines. The vital statistics of all three moles are tabulated in Table 3.

The next task for V Mole 3 in the Saar district will be to bore a major shaft from the surface to a depth of 1.25 km. This will obviously be the most critical task that V Moles have been called upon to undertake, but operational experience to date gives cause for confidence in the outcome.

Reference was made earlier to improvements built into the later versions. Key points are summarized below.

- The method of steering with external guide plates sliding along the shaft walls was dropped in favor of 6 new horizontally mounted cylinders fixed to internal and external kellies, making steering much easier and more precise in zones of weak ground.
- The original steering method using four lasers on separate targets was changed to a single, self-aligning, central laser with fiber-optic transmission to a screen on the driver's control desk, which he can continuously monitor and adjust to keep the machine continually within 12 mm of the true vertical shaft center line.
- The original set of four gripper pads was increased to six pairs thus reducing the unit thrust pressure on the strata. To avoid inducing overbreak, and to ease passage through weak zones of rock such as coal seams or clay bands the six sets of grippers have been vertically duplicated, making it possible to be selective where the grippers are applied. The improvements have been well worthwhile, and overbreak has been almost eliminated.

Table 2—Shafts Bored with V Mole 2 (5-6.5 m diam)

Mine	Firm	Year	Depth m	Advance m/d	Time days
Ibbenburen	Preussag	1977	467	10.6	44
Gottelborn	Saarberg	1978	414	13.8	30
Victoria	Ruhrkohle	1978	106	7.0	15
Walsum	Ruhrkohle	78-79	285	10.5	27
Prosper	Ruhrkohle	1979	180	10.0	18
Erin	Ebv	1979	200	11.0	18

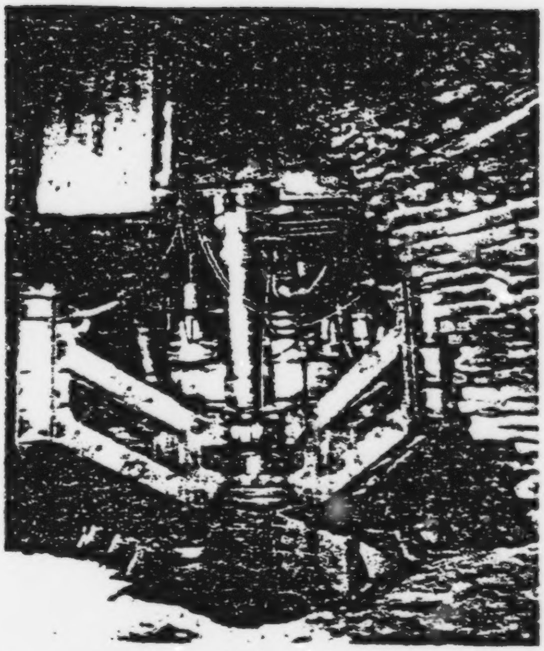


Fig. 3—Starting drilling at Minister Stein with V Mole 1

- As mentioned before, the hydraulic drive has been replaced by straight forward gearing from constant speed electric motors, with improvements in maintenance and downtime.

In opting for economic reasons for the down-the-hole V mole the German contractors didn't neglect the potential of the raise borer, which they continued to use with some excellent results. Reference was made earlier to the paradoxical situation in which the raise-bored hole is completed with minimal equipment and very few men in a short space of time. Problems with putting in the lining, headframe, shaft top doors, lining platform, platform hoists, concreting plant, and shaft formwork meant all the advantages of time and cost were lost. This serious drawback was overcome in a two-pronged attack and was made possible by the use of a new material developed by Pilkington in the UK. Glass fiber-reinforced cement (which experiments proved could also be made into fireproof, shatterproof, rotproof panels of considerable tensile strength) proved ideal for use as a prefabricated shaft lining.

The economic installation of the segmental lining was resolved in a very practical way, by using the stabilizer frame of the cutter head, the entire drill string, and the raise bore machine. The cutters were removed and a steel deck was bolted to the frame of the raise borer head. A working deck, capable of being jacked to give a small degree of independent vertical travel was fitted above this frame, and by extending the drill string rod by rod, the work deck could be lowered from the top

Table 3—Statistical Data on Shaft Borers

Machine	V Mole 1	V Mole 2	V Mole 3
Drive	Elec Hydr	Elec Mech	Elec Mech
Drill diam	4.5-5.0 m	5.0-6.5 m	6.5-8.0 m
Cutter head	3 × 75 kW	4 × 110 kW	6 × 110 kW
Total kW	230 kW	490 kW	745 kW
Torque mkp	27 000	75 000	41 000
Thrust	300 Mp	550 Mp	740 Mp
Load/disc	App. 4 Mp	App. 12 Mp	App. 10 Mp
Cutters	Two discs	One disc	One disc
Disc spacing	48 mm	70 mm	70 mm
Head angle	30°	45°	45°
Gripper pressure	12 kp/cm^2	12 kp/cm^2	12 kp/cm^2
Weight	App. 100 Mp	App. 150 Mp	App. 227 Mp

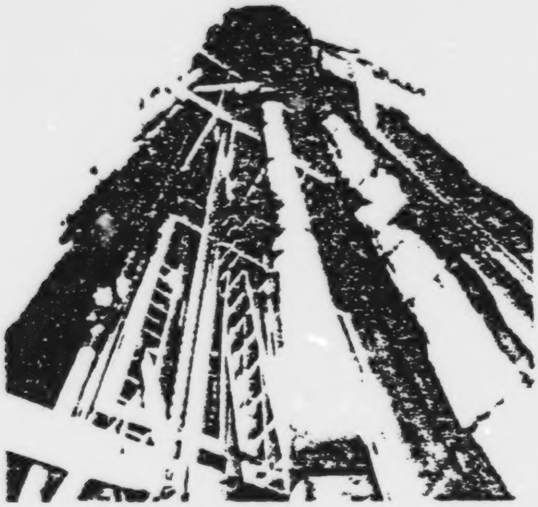


Fig. 4—Shaft lining, guides, and ladderways installed from workdeck lowered by raise-drill string.

of the shaft to the bottom. The panels are equal in height to one drill rod (1.5 m). By minutely adjusting the horizon of the workdeck by the hydraulic jacks, the assembled prefabricated ring could be lifted by the deck (which it stands on) to its required installation horizon.

The lining is only a few millimeters thick, and after location and bracing, it is locked in position by injection of a few centimeters of cement grout, brought down by shaft grout lines and injected to fill the void behind the GRC liner. Catwalks above the workdeck enable bracing from the previously installed ring to be leapfrogged down to the next position. All that is needed at the shaft top is a very simple man hoisting facility and a small hoist to lower the very light GRC segments. Catwalks, ladderways, intermediate decks, pipes and cage, or skip guides are installed at the same time, so that when the head with the workdeck reaches the shaft bottom, a fully equipped shaft is left behind; the cutter head is then taken off and the drill string pulled up through the shaft by the raise borer. The procedure is simple, economic, requires minimal manpower, and is ideal for ventilation requirements. However, there still remains the fundamental problem related to limitations on the amount of torque transmitted by the drill string to the cutterhead.

Something has been done about verticality, where the rule of thumb for achievable accuracy is roughly 1 horizontal to 100 vertical. When the pilot hole is drilled, a lot can be done by using stiffened assemblies and stabilizers behind the cutter, and by limiting the applied thrust and using previous experience in differently dipping formations. But until recently, no guarantee of success could be given.

In Germany, as in the US, in cases where guaranteed accuracy has been demanded by the owner, the pilot hole has been drilled by a separate rig using standard freeze hole drilling techniques. The hole is surveyed every drill joint length and deviated whenever necessary to remain in the target area. In Alabama, using the Robbins RB7 Sp raise-borer, Thyssen drilled the 537-m-deep pilot hole using controlled drilling methods to keep the hole within a 0.5 m radius target area.

The Kind-Chaudron system has been used to drill shafts for almost a century and is still occasionally used today. The Honigman process was used to drill and float in the shaft lining at the Dutch State Emma mine from 1948-1951, and also to construct the two Beatrix shafts to a depth of 546 m between 1954 and 1960. These shafts were built in heavily water-bearing strata and incorporated very sophisticated water-tight linings.

The potential of this system has been more fully developed in the US. Today, large machines with matching hardware are capable of drilling and casing blind shafts to considerable depths. An early example was Shaft Driller's twin masted rig with 450 Mg lifting capacity, which drilled at 5 m diam and installed a continunously welded 4.4-m-diam steel casing, fully fitted with skip and cage winding compartments to provide, when pumped out, an instant shaft 240 m deep. A slimmer, but much deeper shaft was constructed by Fenix & Scisson as an access shaft for underground cavern construction. It was cased at 1 m diam to a depth of 1 km.

A Hughes reverse circulation rig was used more recently by McKinney Shafts Inc. to drill the 200-m-deep shaft at 3.5 m diam at the Emerald #3 mine in West Virginia. The Conoco pilot shaft at Crown Point in New Mexico, drilled to 650 m in the second half of 1980 by Challenger Drilling, is a good example of the high degree of professionalism now being achieved. A 2.13-m-diam, fully hydrostatic steel casing was installed to meet a very tight construction schedule.

The most recent arrival is the combination shaft drill series 300, from Hughes Tool Co. Hughes is marketing this machine as part of a comprehensive shaft construction system, with a capability of 600 m depth at 6-m-diam lined shaft, or deeper with lesser diameter, or to give a greater diameter with a lesser depth. The first shaft to be drilled by the CSD 300 will be constructed in Western Australia by the consortium headed by Santa Fe Engineering and Construction.

For greater depth coupled with bigger diameters there are still the serious constraints referred to earlier: vertical accuracy, the amount of torque that can be applied through the drill string; risk of collapse of the shaft walls; and the rig set-up and time needed to install the lining, pump out the shaft, and make the break-outs.

In the meantime, it will be recalled that the V Mole 1 machine was taken out of active service when V Mole 2 took over in 1977. Some time previously, engineers at Siemag, from Siegen, south of the Ruhr Valley, came up with a practical system of transporting run-of-mine material horizontally and vertically by pipeline using a dual pressure hydraulic circuit. This is currently being used very effectively at the all hydraulic Hansa mine. At the same time, engineering specialists in the

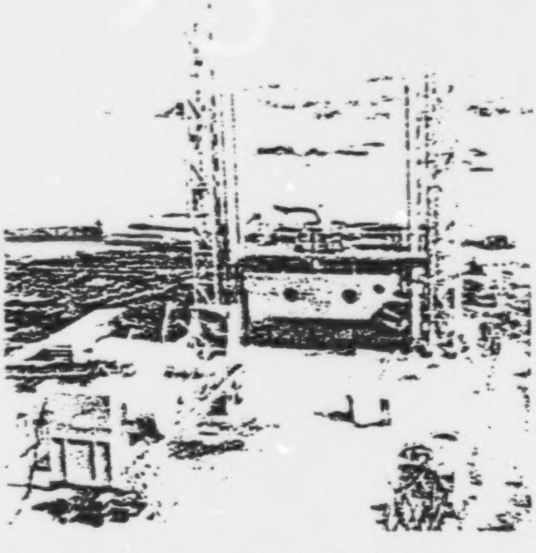


Fig. 5—Hughes combination shaft drill series 300.

German coal industry began to seek a solution to down-the-hole shaft drilling without a pilot hole. An experimental project was put together: the mining contractor supplied V Mole 1. Ruhrkohle provided the shaft and sponsored the trials in a joint venture with Siemag, with the intention of producing a practical and economically viable system. A site for the shaft was selected, and the down-the-hole borer was equipped with a new set of internal works and linked with the dual pressure Siemag closed hydraulic circuits all coupled up to the surface. There were no skips for muck winding, and everything was geared to suck the cuttings into the system and pump them to the surface.

This experimental project was discontinued for further review, and the remaining shaft was bored by the V Mole, using a pilot hole as before. Considerable attention has been given to the outcome of this experimental project, and the debris cutting and removal systems have been re-examined.

Engineering techniques and equipment have been modified due to the practical lessons learned from this project and an order has been placed for a new V Mole to be built by Wirth. This dual purpose machine will operate as either a pure blind shaft borer, or as before, using a pilot hole where underground access is available. In Alabama, the US Department of Energy has terminated a parallel exercise at a mine site owned by US Steel. Here a Robbins down-the-hole rodless borer, equipped with a mechanical mucking system raised the cut material by bucket elevator to deposit it into the shaft skip hoisting system.

Another bored shaft is being constructed for Jim Walter Resources Inc. at a location near Tuscaloosa, AL. Thyssen's V Mole 4 has already finished the first of a series of eight 600-m, concrete-lined shafts of 6.7 m diam. where rates of 18 m/d of finished concrete lined shaft are opening up new dimensions in shaft construction capabilities.

Drilling and mucking equipment produced by today's manufacturers is up-to-date, powerful, and very efficient, but there are undeniable problems associated with noise, dust, and the shortage of skilled manpower. These problems are becoming more difficult for mine construction firms to absorb as environmental, health, and safety legislature requirements are progressively upgraded.

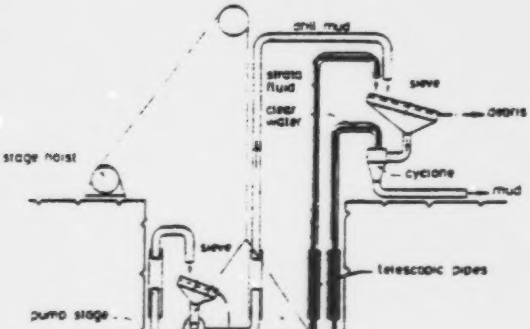


Fig. 6—Shaft full face borer - no pilothole. Schematic Layout of hydraulic muck disposal system.

Fig. 7—Erection of V Mole at Jim Walters #7 shaft in Alabama showing assembly of main drive motors and anchor cylinders.

It is predictable, however, that as long as drill and blast remains the cheapest way to "put a round of muck on the floor," conventional operations will still hold their own, and this is likely to be the case for some time.

With the V mole operations, experience has shown that the shaft walls remain smooth and overbreak is almost eliminated. Therefore, the total volume of shaft-bored rock is at least 15% less than the amount for conventionally sunk shafts. Considerably fewer men are needed than for a comparable size conventional shaft. Dust is removed as it is formed, dragged down the borehole with the suction created by falling chips of rock. The shaft boring technique is at least 55% faster than conventional methods, and this is an important consideration when the shaft lies on the critical path of a major project and when heavy capital investment has been incurred.

Fully lined, smooth-walled shafts can be constructed in remarkably short periods of time. However, independent access to the shaft bottom is a must, and this will remain the biggest permanent limiting factor on the use of this equipment in its present form.

Techniques and equipment for raise boring have been developed to a remarkable degree, enabling shafts to be drilled and reamed out in a very economic manner, assuming good equipment utilization. It is difficult to predict how existing hardware such as the Robbins raise drill model 121 R—the most powerful raise drill in existence, with almost one million kg m/s of thrust—can be improved on. However, the raise borer, coupled with glass reinforced-fiber cement lining techniques described, can be regarded as a very useful and economic piece of equipment with some limitations. There's no doubt it will

Fig. 8—V Mole in action at Jim Walters #7 shaft Alabama, showing gripper/steering pad thrusting against smooth cut profile of shaft wall.

continue to fulfill the very useful role in mining and civil construction that it enjoys today.

A big question mark arises regarding the continued upgrading of shaft drilling machines designed for blind-boring from the surface. As with raise borers, there is no constraining factor on the horsepower of the drive unit on the surface—the limiting factor is the drill string. Strigent health and safety requirements related to dust and ventilation mean that deep mines in the 600 m plus range must have shafts of 6 m diam or more. Machines of adequate capacity to construct such shafts could prove so expensive to operate that they may only be used in cases where the value of the mineral and the urgency of exploiting it are of overriding consideration.

However, the drill string could be replaced by a nonrotating extensible umbilical cord, providing a means of pumping debris to the surface with a power and communications channel for the receipt of signals controlling steering and thrust to a down-the-hole V mole, then all the constraints are removed. Really adverse ground conditions would need to be treated as with conventional shafts today. This could be accomplished by grouting, either pre-grouting or undertaken from the V mole when it reaches the waterbearing measures, or by standard ground freezing techniques throughout the affected horizon, coupled with the installation of a full hydrostatic tubbing lining through the water bearing measures. In most cases, simple inflows from water-bearing strata will be pumped out with the muck and afterwards be handled by the permanent mine pumping installation.

This machine, the direct descendent of V Moles 1, 2, and 3, will have the experience of its predecessors built into it and will bore shafts at whatever diameter and depth the mining operation dictates, without constraint. The depth will be limited only by the self-sustaining capability of the hoist ropes, which move men and materials conveyances from the shaft bottom to the surface.

Conversions

Convert From	To	Divide By
m	ft	0.3048
m^3	bbl	0.159
mm	in	25.4
kW	hp	0.7457
g	troy oz	31.1

Impact Breakers Improve Productivity

he San Manuel mine of Magma Copper Co. is located approximately 45 miles northeast of Tucson, Ariz. The town of San Manuel, the concentrator, smelter, electrolytic refinery and continuous rod casting plant are located eight miles east of the mine site.

Current mine production is in excess of 64,000 tpd. The ore necessary for this large tonnage is produced by a full gravity block caving system. As the undercut of a given block is established, the ore is drawn from the caved area via the draw raises on the grizzly level. It is passed through grizzlies with 14-in. openings into transfer raises to await the loading process on the haulage level 60 ft below.

Production from areas of more competent, hard ground, especially during initial undercut and early production, can present difficult problems in extraction, often resulting in up to a 65 percent reduction of the normal draw rate and an increase in ground weight. The normal procedure in hard, competent ground areas would be to increase manpower and the use of explosives. The excessive use of explosives quickly results in premature damage to the concrete lined grizzly drifts and draw points. As can be seen, the problems rapidly pyramid: oversized hard muck, insufficient draw extraction, excessive weight, manpower increases, increased explosives use, concrete damage and increased number of repair crews.

In an effort to cope with the extraction problems in hard competent blocks, a project of testing high energy impact hammers was begun. A high impact hammer was mounted and tested over a development grizzly pocket in 1973. The results were encouraging enough to initiate the development of a mobile unit in conjunction with Joy Manufacturing Co. Early in 1976 the first prototype unit was placed into production for testing. In late 1976 the prototype machine's performance was promising enough to justify the purchase of three modified units.

Hammer delivers 200 blows per minute

The mobile rock breaker consists of a crawler-mounted chassis, a hammer manipulator boom, a Joy 206 Hefti hydraulic hammer and the necessary power pack and controls.

The air-powered units are 65 in. tall, 43 in. wide and 150 in. long with the boom in the traveling position. The units have an outside turning radius of 70 in., a vertical reach of 11.25 ft, 200° lateral boom swing, and a horizontal reach of 10 ft. The unit's weight is 10,300 lb and weight distribution is such that outriggers are not required in the operation of the boom and hammer.

The hammer weighs only 350 lb and is mounted to the carrier via a three-piece boom assembly. The hammer can deliver up to 200 blows per minute at 1000 ft-lb per blow. This energy is developed by a nitrogen-driven piston. The hammer is capable of 65° crowd in the plane of the boom and 270° of rotation.

Normal procedure for maintaining ore production from the grizzly level in hard, competent ground at the San Manuel mine in the past would be to increase manpower and the use of explosives. Excessive use of explosives often results in premature damage to the concrete-lined grizzly-drifts and draw points and in pyramiding problems

This being the most complex equipment we have on the grizzly level, our challenge was to learn to operate, care for and maintain these machines. Mine operations, maintenance personnel and the mine instruction department developed a procedural manual.

Mine personnel were trained until we had two operators for every machine per crew.

Operators are responsible for minor maintenance, clean-up, proper oil levels and air hoses. Operators also maintain a log book for each machine which records clock hours, down time, major problems, repairs and parts replaced, and the time when maintenance personnel were notified.

The mobile rock breaker now in use on the grizzly level is an air-powered machine weighing 10,300 lb with a hydraulic hammer which can deliver 200 blows per minute at 1000 ft-lb per blow. The breaker can extend into the draw raise for demolition of large boulders and free an area for production in a matter of minutes

Mining system modified for prototype testing

The prototype unit, placed in production in early 1976, was slightly smaller than the units in use today (width 40 in., height 48 in., and length 11 ft) and employed an electrical hydraulic system. The unit was tested in two different areas of the mine. In each area the current mining system was modified to accommodate the mobile rock breaker. In test area #1 the grizzly drift width was increased from 48 in. to 60 in. In test area #2 a block was designed to accommodate the prototype unit. Major design alterations included offset draw raises which provided a grizzly drift with a solid floor for tramming the MRB.

	Tonnage			Explosives		
Location	MRB tph	Control* tph	Ratio MRB/ Control	MRB Tons/Lb Exp.	Control* Tons/Lb Exp.	Ratio MRB/ Control
Test Area #1 26 (7–8) May to Aug. 1976	85	74	1.15	38.5	28.9	1.33
Test Area #2 59 (13–15) Sept. to Dec. 1976	73	48	1.52	12.5	7.7	1.62

*Control—Areas immediately adjacent to the test site containing similar ground rock and which were drawn using manual techniques.

The results of the two test areas, (see table at bottom of page) were encouraging enough to warrant the purchase of three additional improved units.

The modifications on the newer units included increased size for easier maintenance, relocation of grease points for improved accessibility, improved protective plates, and improvement of electrical-hydraulic systems.

During 1977 our major difficulties were caused by insufficient training of both maintenance personnel and operators, contamination of the hydraulic system and inadequate testing facilities.

In early 1978, in an effort to increase productivity of the mobile rock breakers, an intensive training program was instituted for mechanics and operators. In addition, an underground shop was constructed to repair and test hydraulic components. Qualified maintenance personnel were made available on call each shift.

During 1978 the electrical-hydraulic system continued to require extensive maintenance, and in mid-1979 the prototype unit was converted to an all hydraulic system. The hydraulic system proved much superior, and the remaining three units have now been converted to all-hydraulic systems. Improvements are also being made in the filtering systems, bushings and the hammer tilt weldment pin.

Breaker evens out rates of draw in new blocks

Our standard method of grizzly level production (moving ore from draw raises into transfer raises) utilizes basic hand tools: a 16-lb hammer, 5-ft scaling bar, and a grizzly hook.

New production areas require secondary blasting twice each shift to eliminate large or jammed boulders in the throat of the draw raise, and on top of the grizzly in order to maintain production.

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proper heap building and agglomeration techniques at Candelaria would have enhanced the rate of silver extraction substantially. The large 5-ft diam column tests have confirmed it.

Heap leach testing requires considerable amounts of ore—anywhere from 1.5 to 10.000 tons depending upon the amount of assurance that management wants.

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After the initial undercutting of a new block, an even rate of draw is required to establish a proper caving action. Uneven draw not only affects productivity but can cause excessive ground stress, resulting in concrete failure, increased repair, reduced extraction and premature dilution of the ore.

The mobile rock breaker, with its boom and hammer, permits good extension into the draw raises, allowing for the demolition of the large boulders. Large boulders are also broken on the grizzly rails. Thus, in contrast to conventional methods for drawing ore, the mechanical rock breaker will free an area which would be out of production for half of a shift in a matter of minutes.

Our major application has been to use these machines to break in new production blocks and to avoid the problems associated with an insufficient or uneven rate of draw. After proper caving action has been established at 20 to 30 percent tonnage extraction, the conventional method out-performs the mobile rock breaker and the machine is moved to another block.

The mobile rock breaker has proven itself in other applications underground. The machine has been used to moil concrete in the repair of haulage track, the moil hitches for repair sets in concrete supported drifts and, to moil drift floors that had lifted up in squeezing ground areas. These applications all resulted in substantial savings of manpower.

The units have also been used to moil hitches for repair sets. This job normally requires one manshift but can be accomplished by the machine in 45 minutes.

Mechanical availability of 83 percent attained

During the interval from 1977–1978 the mobile rock breaker averaged 54 tons of ore moved per operating hour. This compares to an average of 38 tph produced by conventional methods.

During the interval from 1979–1980 the machine averaged 56 tph vs. 31 tph for conventional drawing of ore. This increased effectiveness of the mobile rock breaker can be attributed to the modifications which were incorporated into the machine as well as the increased draw efficiency which the machine shows in more competent blocky muck found in recent undercuts on the 2600 level.

The mobile rock breaker has a mechanical availability of 83 percent.

There are three basic advantages for using mobile rock breakers on the grizzly levels: blasting is reduced, draw efficiency is increased, and operation of the machines is safer than drawing ore by manual methods of breakage.

The amount of blasting is reduced by the use of the mobile rock breakers. Both the quantity of explosives and the time required to blast draw raises is reduced by 50 percent. Productivity can continue during the time that would be required to be set aside for the normal blasting cycle. The reduction in the use of explosives results in less damage to the draw raise throats from secondary blasting.

In new undercut blocks, tons per manshift have been increased by 50 to 100 percent. By increasing draw efficiency, a more even draw is obtained. This reduces the damage of grizzly lines due to ground weight and decreases the time required to establish a normal caving action.

In four years of operation, no major accidents have occurred to operators, primarily due to reduced exposure to conventional draw hazards.

In our operation, we frequently encounter working areas that, as a result of squeezing ground, become too small for the mobile rock breaker to enter. The deformation of working areas limits the accessibility of the mobile rock breakers.

The most frequent cause of down time in recent months has been caused by hydraulic hose and fitting breakage.

The hammer tends to require major servicing about once a month.

In conclusion, the mobile rock breaker, since introduction of the prototype model in 1976, and after going through extensive testing and modifications, and based on production studies, promises to be a valuable underground mining tool. It has provided faster and more even draw from new production blocks. It has reduced the need for secondary blasting, saving explosives and placement time. It has reduced blast damage and the need for major repairs. This versatile machine, with an appropriate preventive maintenance program and qualified operators, can be a significant improvement over the use of explosives and manual labor.

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MINE WIDE MONITORING SYSTEM AT SAN MANUEL

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ABSTRACT

Magma Copper Company's underground mine at San Manuel, Arizona has grown to the extent that much of its vital remote equipment cannot be monitored effectively. Power distribution, rectification, pumping, ventilation, and compressed air systems have expanded so greatly that problems may develop and become serious enough to adversely affect development and production before being detected.

To minimize potentially costly consequences, a mine-wide data telemetry system is being installed. The system gathers and displays the continuing status of all major electrical switchgear, rectifiers, pumps, ventilation fans, and air compressors. With this information available at one central location, developing problems are recognized quickly, personnel dispatched immediately, and corrective action initiated. In this manner, most "panic" situations and disruptions of development and production can be averted.

In addition to monitoring equipment status, analog data is gathered. Underground airflows and gas levels are monitored to provide valuable information for fire prevention and fighting. Data on compressed air pressures and pumping system water flows aid in the regulation and efficient use of these systems.

Data acquisition is achieved through input/output (I/O) interface cards at microprocessor-based communication terminals. Information is checked, addressed, and then transmitted via frequency shift keying (FSK) modems on twisted, shielded wire pairs.

The remote system is made up of 28 remote terminals located up to two miles from the central terminals on ten different mine levels. Four remotes report to each of seven independent central terminals. By this division, one malfunctioning terminal will affect only a fraction of the total system.

Equipment ON/OFF and protective shutdown status will be displayed by indicator lamps on a 288 square foot graphic panel located in a control center. The graphic display was designed in-house and may be reconfigured easily to keep pace with the continual change and growth of the mine.

INTRODUCTION

Magma Copper Company's San Manuel Division is located in the southeastern part of Pinal Country, Arizona, approximately forty-five miles northeast of Tucson. Seven headframes and the prominent Red Hill serve as landmarks for the mine area.

The concentrator, smelter, sulfuric acid plant, electrolytic refinery, administration building and other plant facilities are located seven miles southeast of the mine area adjacent to the community of San Manuel.

The mine was placed in production in January, 1956, following completion of five shafts and over twenty miles of drifts. Since then, three shafts and hundreds of miles of drifts have been added to raise production from 30,000 tons per day in 1956 to the current level of 63,700 tons of copper ore per day.

Ore is mined from the San Manuel ore body using the block caving method. Caved ore flows down draw raises to the grizzly level where it is sized. The ore then continues to flow into transfer raises to the haulage level 60 ft. below. Here ore cars are loaded from the transfer raises and hauled to the dump pockets by 22-ton trolley locomotives. There are currently three such haulage levels in production.

Four automated, double drum hoists lift the ore from the loading stations to the surface. Each hoist lifts 22 to 28.5 tons of ore every 90 seconds. Nine service hoists operate cages for personnel and supplies.

Mine ventilation is provided by 200 to 450 horsepower axivane fans located on various levels. Fresh air flows at a rate of 1,255,000 cubic feet per minute (ft.^{3}/min.), down three service shafts to the level fans and exhausts up the four production shafts.

Pumps ranging from 400 to 900 gallons per minute (gal./min.) capacity pump water from the mine at the rate of 6,500 gal./min. The mine water is collected by small air or electric pumps and pumped to large sumps on each level. The large pumps relay the water out of the mine in approximately 1,000-ft. lifts. The water is then pumped to the mill, where it is used in the processing circuits.

A total of fourteen compressors are used to deliver 53,632 $ft.^3$/min. of compressed air at 100 pounds per square inch to the mine. The compressed air is used for development equipment, ore flow control through pneumatic actuated doors, spot ventilation, and water collection, using small air-operated pumps.

Electrical power enters the mine's main substation at 115 KV, is transformed to 46 KV, and distributed to various secondary substations on the surface. The primary utilization voltage of 2,400 VAC is then distributed to the surface and underground equipment and transformed for lower utilization voltages. 500KW rectifier stations underground supply the 270 V.D.C. power for the trolley haulage system.

The 2,150 hourly employees and 340 salary employees who work at the San Manuel mine have the distinction of working for one of the world's largest underground copper mines.

SYSTEM

An upset to any of the vital support systems in the mine can have a detrimental effect on both production and development. As the mine continued to expand and the support systems grew, it became obvious that a method was needed to effectively monitor these support systems. Several alternatives were explored, objectives were set, and criteria determined for the design of a system.

The objectives of the system were to reduce production and development losses, lower maintenance and operating costs, and provide a central control location for aiding in fire control and personnel protection. These objectives were accomplished through a system capable of continuously gathering pertinent data and equipment status into one central location so that developing problems could be quickly recognized and corrective action initiated.

The system criteria calls for minimum cable installation, a flexible and expandable configuration, high noise immunity, and optimum reliability in environments of 100% humidity and 50°C. Ease of interfacing at both the process and graphic display, system simplicity, and preprogrammed functions are also considerations.

A hard-wired remote monitoring/control system was rejected because 1,500 to 2,000 pairs of wires would have been necessary. The job of installing cables and moving associated equipment to new locations as mining progressed would have been extremely large. In addition, maintenance and installation costs would have been prohibitive with this type of system.

Multiplexing of data onto common transmission lines greatly reduces the cable pairs required to collect the data necessary and transmit it to a remote monitoring location. Point to point multiplexing equipment requires a large investment in capital and sufficient space. Multiplexing by itself does not fill the requirements of a 2,000-point telemetry system. All criteria were satisfied, however, by the Slo Electronics system which was selected.

The system designed for the mine is a programmable system which transmits data over a multiplex system and functions as an "intelligent network." The substations, Remote Control Terminals (RCTs), and the Central Communications Terminals (CCTs) are "intelligent," programmable units which transmit processed data only when a change occurs in a process. All conventional control room functions, such as equipment status, indicating lamps, alarms, control commands, and analog signal transmissions, are provided by the basic system.

The basic system configuration is one CCT for every four RCTs. Every I/O point connected to a RCT corresponds to an I/O point on the CCT. Connections are similar to a point to point system, making installation simple and straight-forward. The four RCTs to one CCT configuration was adopted for high system reliability and short RCT/CCT response time.

The process is always connected to the RCT, which maintains an image of the process in its memory by reading the signal inputs at short intervals. When a signal changes, the status of the measurement point involved and the time of the event are transferred to a transmission buffer.

CCTs control the data transmission between the control room and the substations. Each CCT polls its RCTs for any "new" information at the transmission buffer. Data is transmitted via time division multiplexing in the form of FSK pulse coded modulation.

Since the number of RCTs per CCT is limited to four, and very few changes occur in normal situations, the information is transmitted with a very short delay. Priority status is arranged to transmit control messages from the CCT to the RCT first, and all data is transmitted as soon as it is practical. Each message has a data block length of up to 16 bytes and as many as 24 check bits, which ensures efficient data transmission security.

The structures of the CCT and the RCT are almost identical. The CCT transfers the data from the RCT's (status indications and alarm measurements) to its interface units which drive lamps, LED's, meters, or recording devices. The CCT indicates changes in status and alarms by flashing appropriate indicating lamps. The rate of flashing indicates the current state of the status change or alarm. An audible tone is sounded when a new alarm condition is received by the CCT.

Control switches installed at a central console issue commands to the CCTs for internal and remote control functions. Resetting flashing lamps, alarm acknowledgements, lamp testing, and signal blocking are all internal CCT functions. Remote control of equipment is accomplished via CCT to RCT message transmissions (see Figure 1).

The CCT's are interfaced to a Magma designed and built 288 ft.^{2} graphic display panel. The ever moving and expanding mining operations meant that the panel graphics and indicating lamps had to be designed so that they could be easily reconfigured. The display structure was built with unistrut which supports framed, triple plate glass. A graphic depiction of the mine was laid out on a series of 2-ft. by 4-ft. white acrylic panels (see Figure 2). The graphics were designed in a process flow arrangement utilizing colored graphic tapes and symbols. Aluminum grids were fastened to the back of the acrylic panels, facilitating the installation of indicator lamps. The completed panel assemblies were aligned behind the glass and locked in place by spring loaded bars.

Interfacing between the 16-channel interface cards and the graphics is achieved through a wire wrap interface board. Sixteen conductor flat cables are brought from the interface cards to the board and plugged into headers with wire wrap pins that extend through the board. Forty-conductor cables with wire wrap headers at each end connect the interface board to the graphic panels. Proper interconnections are made on the wire wrap pins at the interface board and the indicator lamps in the panel grids are then connected to the 40-conductor cable headers. This arrangement makes removal of individual panels possible by simply unplugging the 40-conductor cable, removing the tension bar, and taking the panel down. Adjoining panels are, therefore, not disturbed when a panel is removed for modifications.

Control inputs to the CCT's are through pushbuttons on the control console. These inputs start and stop primary fans, rectifiers for trolley power and any other equipment deemed necessary to be placed under remote control. All alarm resets and test buttons are also located on the control console.

A brief discussion of the areas and equipment monitored is necessary to obtain an overview of the system. The scope of the remote monitoring/control system includes status of power distribution switchgear, haulage rectifiers, pumps, ventilation fans, air compressors and the chilled water system. Analog information is gathered from remote points, indicating air flows and carbon monoxide levels, compressed air pressures and water flows in the pumping systems. Over 300 individual pieces of equipment are monitored from 26 remote locations.

One of the largest systems to be monitored is the mine dewatering pump system. Over 120 100-HP to 350-HP pumps lift approximately 6,500 gallons of water per minute out of the mine. Conditions such as bearing failures and motor overloads could cause pumps in remote areas to shut down unnoticed. Local flooding may occur, which can result in production and development delays. To minimize these potentially costly consequences, the ON/OFF status and maintenance/alarm condition of each pump is monitored. A bearing failure, motor overload or maintenance shut-down is flagged by the maintenance/ alarm point.

Contact closures at the pump motor starters supply the necessary status and alarm information to the nearest RCT. The RCT transmits any changes in status or alarm to the central surface units which decode the information and operate the appropriate indicator lamps to show which pumps are on, off or tripped off.

This same method is used to monitor the mine's potable water system. Insufficient supplies of potable water lead to mining losses. To minimize this occurrence, pump status, sump levels, and automatic valve positions are monitored.

Mine compressed air is another potential problem area. Large surface air compressors supply 100 psi of air through a network of pipes to the mining processes underground. Mining delays are encountered if line pressures fall below 80 psi. Operating status of the compressors and remote line pressures are monitored to minimize these occurrences.

Major electrical power distribution switchgear is monitored for ON/OFF status. This includes the 115 kV, 46 kV and 2.4 kV distribution systems, both on surface and underground. The condition of all circuit breakers and fused line starters for branch circuit protection is observed. Causes and locations of power outages are rapidly determined.

For the purpose of fire control, the status of all main air doors and main ventilation fans are monitored. Additionally, the fans and the release of fire stench into the compressed air lines can be remote controlled from the control center. Trolley haulage rectifiers are another type of equipment which is remote concontrolled due to the potential shock and fire hazards that the trolley system presents.

SUMMARY

Remote monitoring of all the equipment discussed will enable one person to maintain close surveillance of vital functions in the mine. Operators immediately recognize developing problems and dispatch the proper personnel to minimize the interference with operations. The control room centralization is adjacent to the haulage and operational dispatch station which is already operational. Their close proximity provides a vital communication link with the operations of the mine.

The management of Magma Copper Company has invested in a system which ties technology and innovation to the processes involved in an underground copper mine. Future dividends will be provided by increased productivity, safety and reduced cost of operations.

The Authors

C. A. Metcalf has been employed by Magma Copper Company since 1979. He currently holds the position of electrical supervisor, overseeing communications and instrumentation at the San Manuel mine. Previously, Mr. Metcalf was employed by Conoco Inc., as a maintenance supervisor at their Florence Copper Project and then as Maintenance Superintendent at their Conquista Uranium Mill. He has nineteen years of experience as an electrician and a supervisor in the fields of electrical system maintenance, communications and process control instrumentation.

K. L. Billhartz is currently an electrical supervisor at the San Manuel mine, overseeing the installation of the monitoring/control system. He has been in the employ of Magma Copper Company since 1978, originally starting as an engineer in the maintenance planning department. Mr. Billhartz was previously employed in the maintenance operation of Freeman United Coal Mining Company, a division of General Dynamics Corporation. An electrical engineer, he has five years of experience within the mining industry since graduating from Southern Illinois University in 1976.

DATA FLOW SHEET

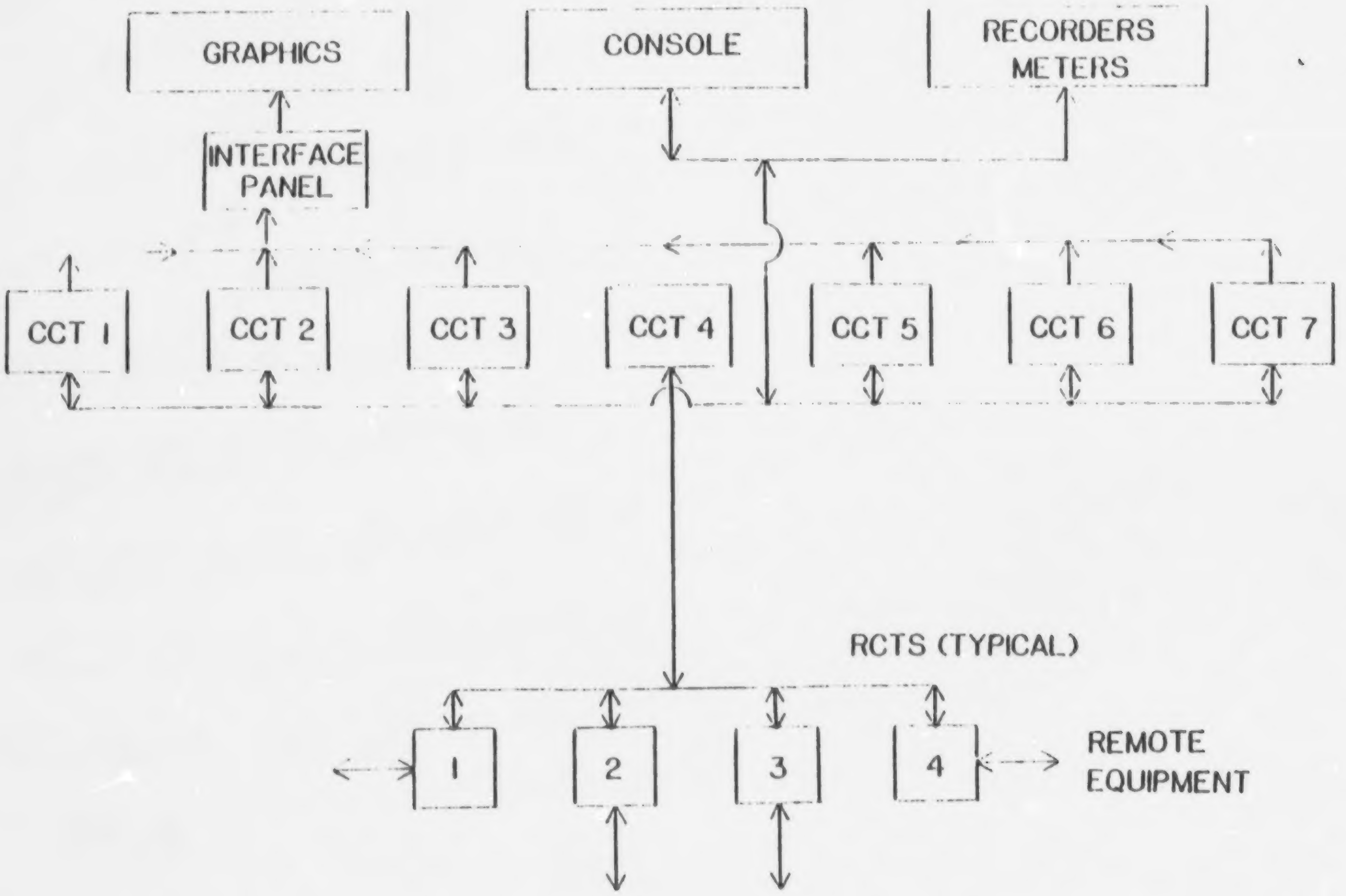
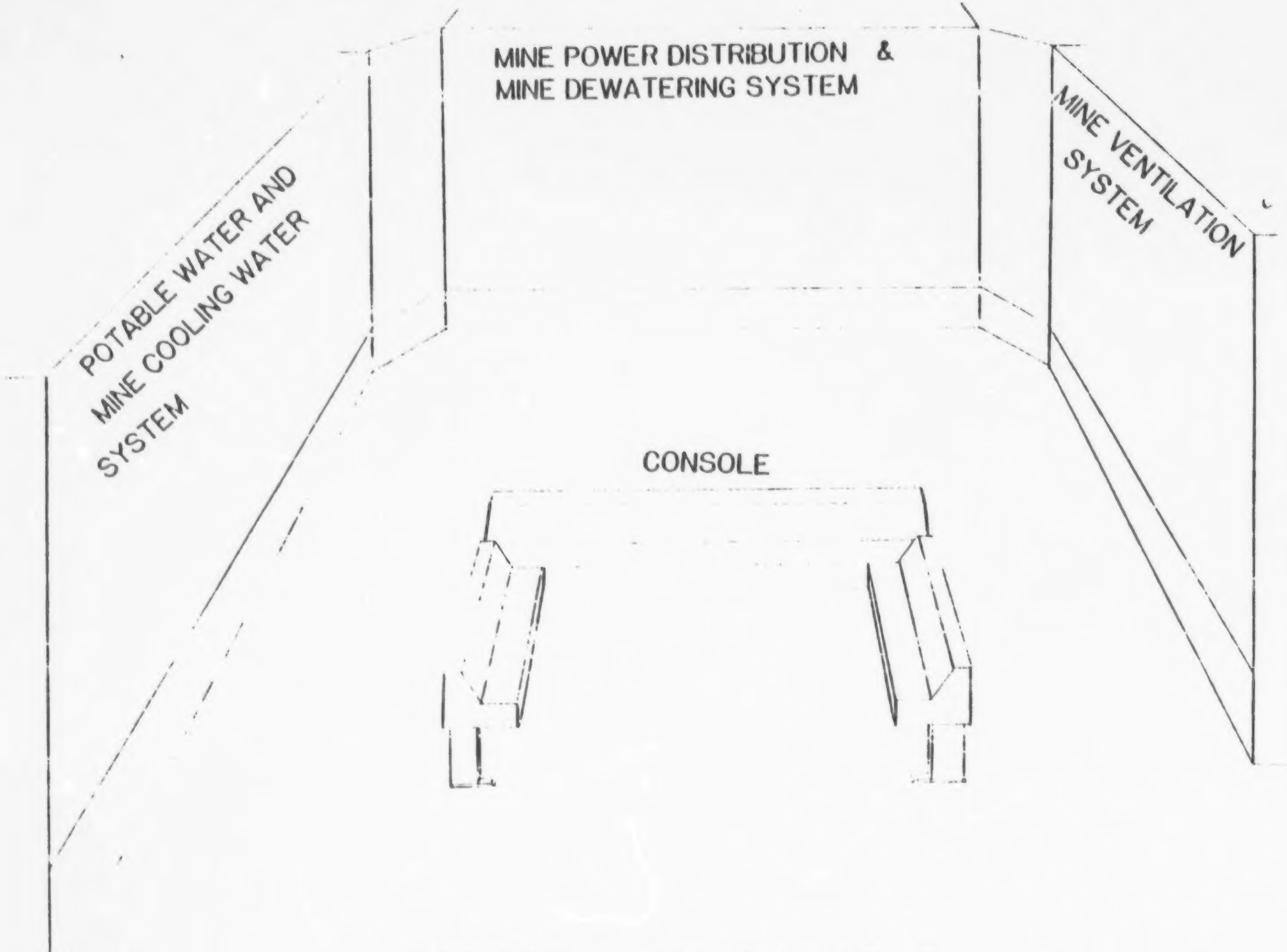

GRAPHICS
CONSOLE
RECORDERS
METERS
INTERFACE
PANEL
CCT 1
CCT 2
CCT 3
CCT 4
CCT 5
CCT 6
CCT 7
RCTS (TYPICAL)
1
2
3
4
REMOTE
EQUIPMENT

FIGURE 1



MINE POWER DISTRIBUTION &
MINE DEWATERING SYSTEM
POTABLE WATER AND
MINE COOLING WATER
SYSTEM
MINE VENTILATION
SYSTEM
CONSOLE

CONTROL ROOM LAYOUT

FIGURE 2

PART 6

CONCLUSIONS

6.0 CONCLUSIONS

It is readily evident that a considerable amount of development is still taking place in mining equipment for the hard rock metal mining industry. Most of this development has the purpose of improving safety and productivity.

The general conclusions that can be drawn from the data produced by the study are:

(a) There is a trend toward the use of more electric powered equipment, which could eventually lead to the development of the all electric mine.

(b) There is a trend to the use of remote control and automated equipment to reduce manning levels and costs and improve safety.

(c) There is a trend towards the elimination of high cost methods and equipment, such as diesel haulage.

(d) There is an obvious trend for developments, particularly within the European civil underground construction industry, to eventually be applied to underground mining.

(e) The bulk of the major developments in hard rock mining equipment are due to the interest of the major mining companies, who alone appear to have the cash flow and capacity to develop new ideas for the eventual benefit of the mining industry.

(f) The data developed by the questionnaire indicates a keen interest in the development of new ideas, but did not generate, nor was it expected to generate, sufficient information on which to determine the timing of the application of those new ideas to mining operations. The timing is more likely to be dependent on such factors as metal prices and interest rates, matters over which mining companies have little or no control.

(g) The need exists to find ways and means for reducing the time required to bring new mines into production. The indications are that new equipment developments are beginning to make such means available to the mining engineer.

(g) We believe that the areas that offer the greatest potential benefit to the Canadian underground metal mining industry are:

i) in drilling - electrohydraulic methods
- remote and automatic control
- ITH drilling

ii) in loading
  - remote control
  - electric powered units
  - continuous loading systems

iii) in transport
  - automatic rail systems
  - electric haulage (LHD, locomotive)
  - crushers and conveyors

iv) in support
  - automated rock bolting equipment

v) in mechanized excavation
  - shaft boring machines
  - tunnel boring machines
  - raise boring machines

vi) in safety and control
  - monitoring systems
  - mechanized scaling and secondary breakage

(h) There is also a need for a more integrated look to be taken at the various elements of the mining cycle to maximize the equipment capacities within each part of the cycle and in relation to the whole system in order to develop the greatest benefit from the many and varied technological improvements that have and are becoming available for use in the mining industry.

(i) More thought needs to be given to retaining waste rock in the mine rather than hoisting it, at considerable expense, to the surface. Systems for disposing of waste rock underground could offer considerable cost savings.

(j) The declared scope of this preliminary study did not include an examination of the engineering and economic benefits of the developments. Such an examination is necessary to properly evaluate the new developments and forms the basis for the second part of the study as set out in Part 7.

PART 7

RECOMMENDATIONS FOR FURTHER WORK (PART II)

PART 7 - RECOMMENDATIONS FOR FURTHER WORK (PART II)

CONTENTS

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7.0 RECOMMENDATIONS

The scope of work submitted by Montreal Engineering for the Preliminary Study was to produce case histories and an assessment of the likely application to Canadian mines of developments in hard rock mining equipment throughout the world.

The in-depth studies of the new developments so identified in the Preliminary Study were to be developed from the findings of that study and undertaken as Part II of the project.

We consider that it would be of benefit to the Canadian metal mining industry that these in-depth studies be undertaken in the following areas:

(a) The potential for developing on all electric mine including computer control of the system to improve efficiency.

(b) The potential for high or continuous capacity systems for loading and transporting ore.

(c) The potential for the application of automation to mining equipment.

(d) The application of high speed mechanized excavation methods to shafts, raises, inclines and tunnels or headings.

(e) The application of monitoring and remote operation systems.

(f) The feasibility of retaining waste rock underground, and the systems necessary to achieve this.

These concepts are somewhat different from those envisaged in the initial proposals, but they reflect the areas of concern and the area of most potential. We will be pleased to submit proposals for carrying out this work upon agreement with CANMET on the subjects to be covered.

APPENDIX A

QUESTIONNAIRE

MINING EQUIPMENT QUESTIONNAIRE

Please check against the appropriate answer and add comments where relevant.

1. Size of Mine.

< 500 TPD ☐
500 - 1000 TPD ☐
1000 - 5000 TPD ☐
>5000 TPD ☐

2. Minerals Mined.

Al ☐
Cd ☐
Ca ☐
Co ☐
Cu ☐
Au ☐
Fe ☐

Pb ☐
Mg ☐
Mo ☐
Ni ☐
Pt ☐
Se ☐
Ag ☐

U ☐
Zn ☐
Sn ☐
S ☐
Potash ☐
Gypsum ☐
Salt ☐
Asbestos ☐

Other ☐ Please Specify.

3. Type of Ore Body.

Massive ☐
Veined ☐
Complex ☐

Steeply diping > 45° ☐
45° - 10° ☐
< 10° ☐

4. Methods of Mining.

Room & Pillar ☐
Block Caving ☐
Panel Caving ☐
Sublevel Caving ☐
Shrinkage Stoping ☐
Open Stoping ☐
Cut & Fill ☐
Undercut & Fill ☐

Sublevel Blasthole ☐
Vertical Retreat Blasthole ☐
Large dia. Blasthole ☐
Benching ☐
Top Slicing ☐
Square Set ☐
Other ☐
Please Specify

5. Mining Equipment Types.

☐ A. Portable Crushers
☐ B. In the Hole Drills
☐ C. Automated or Mechanised Drilling
☐ D. Hagglund loaders
☐ E. Hagglund trains
☐ F. Belt conveyors
☐ G. Chain conveyors
☐ H. Vertical or curved conveyors
☐ I. Full face tunnel boring machines
☐ J. Roadheader type machines
☐ K. Continuous Miners
☐ L. Low profile equipment
☐ M. Pipeline transport of ore, etc.
☐ N. New support systems
☐ P. Remote control of equipment
☐ Q. Monitoring equipment
☐ R. Transport ore, men, materials
☐ S. Safety & Environment
☐ T. Raise Boring
☐ U. Blind Shaft boring

6. Has the use of any of this equipment had a considerable impact on (add Equipment Ref. letter).

Mine Design ☐ ....
Layout ☐ ....
Working Method ☐ ....
Safety ☐ ....
Ground control ☐ ....
Economics ☐ ....

7. Briefly, identify equipment, uses, advantages and cost benefits

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8. APPLICATION

If you consider any of this equipment has or could have wide applications in hard rock mining, please specify by product reference letter below:

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9. Please add any other information, which you may consider useful.

....

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APPENDIX B

ACKNOWLEDGEMENTS

A. MINING COMPANIES

We would like to thank the following mining companies who provided data and comment in answer to the questionnaire issued.

CANADA

Algoma Ore Division - Wawa, Ontario
Asarco Inc. - Buchaus Mine - Buchaus, NFLD.
Bell Asbestos Mines - Thetford, Quebec
Brunswick Mining & Smelting - Bathurst, N.B.
Camflo Mines - Malartic, Quebec
Campbell Chibougamau - Chibougamau, Quebec
Campbell Red Lake - Balmertown, Ontario
Canada Tungsten Mining - Turigoten, NWT.
Cominco - Sullivan Mine - Kimberley, B.C.
  - Con Mine - Yellowknife, NWT
  - Vanscoy Mine - Vanscoy, Sask.
Cons. Durham Mines & Resources - Prince William, N.B.
Dankoe Mines - Keremos B.C.
Darius Gold mine - Cadillac, Quebec
Denison Mines - Elliott Lake, Ontario
Eldorado Nuclear - Eldorado, Sask.
Falconbridge Copper - Lake Dufault Div. - Noranda, Quebec
  - Opemiska Div. - Chapais, Quebec
Falconbridge Nickel - Falconbridge Mine - Sudbury, Ontario
  - Strathcona Mine - Sudbury, Ontario
  - Onaping Mine - Sudbury, Ontario
Heath Steele (Noranda) - Newcastle, N.B.
Hudson Bay Mining & Smelting - Flin Flon Mines - Flin Flon, Man.
  - Snow Lake Mines - Flin Flon, Man.
Meco Limited - Copper Cliffs - Copper Cliff, Ont.
  - Shebandowan - Ontario
Kerr Addison - Kerr Mine - Virginiatown, Ont.
Lamaque Mining Co. (1964) - Val d'Or, Quebec
Louvem - Val d'Or, Quebec
Madeleine Mines - St. Anne Des Monts, Que.
Mattabi Mine - Ignace, Ontario
Niobec Inc. (Les Services TMG) - St. Honare, Quebec
Noranda Mines - Boss Mountain Div. Hendrix Lake B.C.
  - Gaspe Mines - Murdochiville, Que.
  - Geco Div. - Manitonwadge, Ont.
  - Matagami Div. - Matagami, Quebec
Parmour Porcupine - Timmins, Ontario
Potash of America - Saskatoon, Sask.
Potash Corp of Sask. Mining - Cory Div. - Saskatoon Sask.
  - Lanigan Div. - Lanigan Sask.
  - Esterhazy Div. - Esterhazy Sask.
Rio Algom - Elliott Lake, Ont.
Selco Mining - South Bay Mine - Ear Falls, Ont.
Sigma Mines (Quebec) - Val d'Or, Que.

Silvana Mines - New Denver, B.C.
Sturgeon Lake Mine (Falconbridge Copper) - Ignace, Ont.
Tantalum Mining - Bernic Lake, Man.
Teck Corporation - Beaverdell Div. - Beaverdell, B.C.
  - Silverfield Div.- Cobalt, Ont.
United Keno Hill - Elsa, Yukon
Western Mines - Campbell River, B.C.
Whitehorse Copper - Whitehorse, Yukon
Willroy Mines - Kirkland Lake, Ont.

U.S.A.

Homestake Mining Co. - Lead - S. Dakota
Magma Copper Company - San Miguel Div. - San Miguel, Arizona
National Potash Company - Carlsbad, New Mexico
New Jersey Zinc Company - Sterling Div. - Ogensburg, New Jersey
Ozark Lead Company - Ozark Div. - Sweetwater, Missouri
St. Joe Lead Company - Pea Ridgehive - Sullivan, Missouri
Union Carbide Corp. Metal Div. - Uravan Div. - Uravan, Colorado
United Nuclear Corp. Div. - New Mexico Div. - Gallup, New Mexico

AUSTRALIA

Abminco N.L. - Cleveland Mine - Luina, Tasmania
Agnew Mining Co. - Agnew Nickel Project - Leinster, W. Australia
Kalgoorlie Mining Ass. - Mt. Charlotte Mine - Finniston, W. Australia
Mount Isa Mines Ltd. - Mount Isa, Queensland
Mount Lyell Mining & Railway Co. - Queenstown Div. - Queenstown, Tasmania
Selcast Exploration Ltd. - Spargoville Andrews Mine - Kambalda, W. Australia
Western Mining Corporation - Kambalda Nickel Div. - Kambalda, W. Australia

CHILE

Codelco - Chile - Andina Division - Saladillo Chile
  - El Teniente Division - Sewell, Chile
  - Salvador Division - Salvador Chile

EIRE

Avoca Mines Ltd. - Avoca Mine - Arklow, Wicklow Eire
Mogul of Ireland - Nenagh Mine - Nenagh Tipperary, Eire

FINLAND

Outokumpu Oy. - Vihanti Mine - Lampinsaari, Finland
  - Pyhasalmi Mine - Pyhakumpu, Finland

W. GERMANY

Preussag A.G. - Rammelsberg Mine - Goslar, W. Germany

INDIA

Indian Copper Corporation - Mosaboni Mine - Singhbhum, Bihar

JAPAN

Mitsui Mining & Smelting - Kamicka Mine - Yoshiki Gun - Gifu
Shimokawa Mining Co. - Shimokawa Mine - Hokkaido

NORWAY

Elkem - Spigerverket A/S - Norsk Nefelin Mine - Stjernsund
Folldal Verk A/S - Tverr Fjellet Mine - Hjerkinn
Fosdalens Bergverks-ak-Tieselskab - Malm Mine - Trondelag

PERU

Compania Mineral del Madrigal - Madrigal Mine - Arequipa

PHILLIPINES

Apex Exploration & Mining Co. - Masara Project - Maco
Philex Mining Corporation - Santa Tomas Mine II - Benguet

SOUTH AFRICA

Blyvooruitzicht Gold Mining Co. - Blyvooruitzieht Mine - Blyvoorzuitsig, Transvaal
Doornfontein Gold Mining Co. - Doornfontein Mine, Johnnesburg Transvaal
Kloof Gold Mining Co. - Kloof Mines - Westonaria - Transvaal
Libanon Gold Mining Co. - Libanon Mines - Westonaria - Transvaal
Stilfontein Gold Mining Co. - Stilfontein Mines - Stilfontein - Transvaal
Western Platinum Ltd. - Marikana Mines - Marikana - Transvaal

SPAIN

Union Explosives Rio Tinto SA - Potasas Del Llobregat Barcelona

SWEDEN

LKAB - Malmberget Mine - Malmberget
LKF Steel Gruvorvalthingen - Hofors Gruvor Mine - Hofors

ZAMBIA

Nchanga Cons. Copper Mines - Konkola Div. - Chililabombwe
- Chingola Div. - Chingola
Roan Consolidated Mines - Luanshya Mine - Luanshya

B. SHAFT & TUNNEL CONTRACTORS

We would like to thank the following contractors who provided data and comment for the survey.

Guy. F. Atkinson Construction Company - San Franscisco, California
Boskalis Westminster Construction Ltd. - Glasgow, Scotland
Canadian Mine Services - Brampton, Ontario
Frontier-Kemper Constructors - Evansville - Indiana
Harrison Western Corporation - Denver, Colorado
Morrison-Knudsen Company Inc. - Boise, Idaho
M.M. Sundt Construction Co. - Tucson, Arizona
Teton Exploration Drilling Company - Casper, Wyoming
Thyssen Mining Construction Inc. - Lakewood, Colorado
Walsh Construction Company - Darien, Connecticut

C. MANUFACTURERS & SUPPLIERS

We would like to acknowledge the help we have received from the following manufacturers and suppliers.

1. Drilling Equipment

Atlas Copco
Chicago Pneumatic Tool Co.
Compair Construction & Mining
Dresser Industries
Eimco (G.B.)
English Drilling Equipment Co.
Furukawa Rock Drill Sales Co.
Gardner Denver International
Ingersoll - Rand Co.
Joy Manufacturing Co.
Koken Boring Machine Co.
Montabert
Secoma
SIG - Swiss Industrial Co.
Smith International
TRW Mission Mfg. Co.
Tampella AB, OY.
Torque Tension Ltd.
Toyo Kogyo Ltd.

2. Loading Equipment

Anderson Strathclyde
Atlas Copco
Caterpillar
Clark Equipment
Eimco

Eimco (GB)
Elmac Corp.
Equipment Minier
GHH Skerkrade
Goodman Equipment Corp.
Hausherr & Soehne
Joy Manufacturing Co.
Schopf
Toro Oy
Wagner Mining Equipment

3. Transportation Equipment

a) Conveyors

Anderson Strathclyde
Babcock International
Cable Belt Conveyor Inc.
Dowty Meco Ltd.
Flexowall Corp.
Fox Manufacturing Co.
Huwood Ltd.
Kone Corp.
Long Airdox Co.
Realisation Equipements Industriels
Radmark Engineering
Rexnord Inc.
Robins Conveyor Co.
Serpentex Conveyor Corp.

b) Crushers & Feeder Breakers

Allis Chalmers
Eagle Crusher Co.
Goodwin Barsby
Hammermills Inc.
KHD Humbolt Wedag AG.
Kone Corp.
Rexnord Inc.
Sheepbridge Equipment
Stameer Corp.
Westfalia Lunen

c) Rail

ASEA
BEDIA GMBH & Co.
Clayton Equipment
Goodman Equipment Corp.
Greenbat Ltd.
Gyro Mining Transport Ltd.

Hudson Ltd.
Hunslet Ltd.
SIG - Swiss Industrial Co.
Wingrove & Rogers Ltd.

d) Trucks & Utility Vehicles

Jarvis Clark
DJB Engineering
Eimco Mining Machinery International
Eimco GB
Elmac Corp.
Equipment Minier
FMC Corp.
G.H.H. Sterkvade
Getman Corp.
Goodman Corp.
Gyro Mining Transport
Kersey Manufacturing
Kockum Landsverk AB
Mining Transportation Co. AB
National Mine Services Co.
Nitro Nobel MEC AB
Normet SA
Schopf
Teledyne
Wagner Mining Equipment Co.

4. Rock Bolting & Shotcreting Equipment

Aliva Inc.
Atlas Copco
Eimco Mining Machinery International
Eimco (G.B.)
Equipment Minier
Ingersoll Rand
Secoma
SIG-Swiss Industrial Co.
Salzgitter AG
Tamrock Oy

5. Mechanized Drivage

a) Continuous Miners

Jeffrey Mining Machine Co.
Joy Manufacturing Co.
Lee Norse Co.

b) Roadheaders

Alpine Equipment Corp.
Anderson Strathclyde
Dosco Overseas Eng.
Eickhoff Gmbh
Hausherr & Soehne Garbh
Thyssen (G.B.)
Westfalia Lunen

c) Tunnel Boring Machines

Atlas Copco AB
Atlas Copco Jarva Inc.
Calweld Div. Smith International
Dresser Industries
Mannesmann DEMAG
Robbins Co.
Thyssen (G.B.)
Wirth Gmbh

d) Raise Borers

Calweld
Dresser Industries
Ingersoll Rand
Kawasaki Heavy Industries
Koken Boring Machine Co.
Robbins Co.
Smith International
Turmag (G.B.)
Wirth Gmbh

e) Shaft Borers

Calweld
Dresser Industries
Drilco Industrial
Frontier - Kemper Constructors
Mannesman DEMAG
Robbins Co.
Wirth Gmbh

D. TECHNICAL JOURNALS

We wish to acknowledge the extensive use which has been made of the papers published in the following technical journals:

ASEA Journal
C.I.M. Bulletin & Reporter
Canadian Mining Journal & Annual Review

Civil Engineering
Coal Age
Coal Mining & Processing
Colliery Guardian
Engineering and Mining Journal
Inco Bulletin
Gluckauf
Mine & Quarry
Mining Congress Journal
Mining Engineer
Mining Engineering
Mining Journal
Mining Magazine & Annual Review
Mining/Processing Equipment
Mining Technology
Northern Miner
Pit & Quarry
Transactions I.M.M.
Tunnels & Tunnelling
Western Miner
World Coal
World Mining